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THE HOLLEY MEMORIAL,

WASHINGTON SQUARE, NEW YORK CITY.

(*See page xvii.*)

TRANSACTIONS

OF THE

AMERICAN INSTITUTE OF MINING ENGINEERS.

VOL. XX.

JUNE, 1891, TO OCTOBER, 1891,

INCLUSIVE.

NEW YORK CITY:
PUBLISHED BY THE INSTITUTE,
AT THE OFFICE OF THE SECRETARY.
1892.

PHILADELPHIA:
SHERMAN & CO., PRINTERS.

PREFACE.

THE present volume contains the Holley Memorial Address, delivered in New York, October 2, 1890, by Mr. James Dredge, of London, together with an account of the ceremony of unveiling the Memorial, which took place on the same day, and a view of the bust and pedestal. This material was necessarily omitted from Volume XIX., for lack of room. It will not be found to have lost its interest by reason of this delay in publication.

There is also in this volume, besides the usual analytical index, an analytical index covering Volumes XVI. to XX. inclusive. Taken together with the Index Volume (I. to XV.), this covers the entire series of the *Transactions*, and greatly increases their value for purposes of reference.

In order to make room for the special features above named, and to secure at the same time a change desirable in itself, the Volume has been closed with the October Meeting, so that each future volume may begin with the February (Annual) Meeting, and contain the proceedings of a complete calendar year.

MAY, 1892.

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For year ending February, 1892.

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* Appointed to fill the vacancy caused by the death of Mr. J. B. Mackintosh.

LIST OF THE MEETINGS OF THE INSTITUTE AND THEIR LOCALITIES FROM ITS ORGANIZATION TO FEBRUARY, 1891.

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I.	Wilkes-Barre, Pa.,*	May, 1871,	i. 3
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III.	Troy, N. Y.,	November, 1871,	i. 13
IV.	Philadelphia, Pa.,	February, 1872,	i. 17
V.	New York, N. Y.,*	May, 1872,	i. 20
VI.	Pittsburgh, Pa.,	October, 1872,	i. 25
VII.	Boston, Mass.,	February, 1873,	i. 28
VIII.	Philadelphia, Pa.,*	May, 1873,	ii. 3
IX.	Easton, Pa.,	October, 1873,	ii. 7
X.	New York, N. Y.,	February, 1874,	ii. 11
XI.	St. Louis, Mo.,*	May, 1874,	iii. 3
XII.	Hazleton, Pa.,	October, 1874,	iii. 8
XIII.	New Haven, Conn.,	February, 1875,	iii. 15
XIV.	Dover, N. J.,*	May, 1875,	iv. 3
XV.	Cleveland, O.,	October, 1875,	iv. 9
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XXXV.	Boston, Mass.,*	February, 1883,	xi. 217
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XXXIX.	Chicago, Ill.,	May, 1884,	xiii. 1
XL.	Philadelphia, Pa.,	September, 1884,	xiii. 285
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XLII.	Chattanooga, Tenn.,	May, 1885,	xiv. 1
XLIII.	Halifax, N. S.,	September, 1885,	xiv. 307
XLIV.	Pittsburgh, Pa.,*	February, 1886,	xiv. 587
XLV.	Bethlehem, Pa.,	May, 1886,	xv. lxiii.
XLVI.	St. Louis, Mo.,	October, 1886,	xv. lxx.
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LIV.	Colorado,	June, 1889,	xviii. xvii.
LV.	Ottawa, Canada,	October, 1889,	xviii. xxiv.
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LVII	New York, N. Y.,	September, 1890,	xix. vii.
LVIII.	New York, N. Y.*	February, 1891,	xix. xxv.
LIX.	Cleveland, O.,	June, 1891,	xx. lvi.
LX.	Glen Summit, Pa.,	October, 1891,	xx. lxi.

* Annual meeting for the election of officers. The rules were amended at the Chattanooga meeting, May, 1878, changing the annual election from May to February.

† Begun in May at Easton, Pa., for the election of officers, and adjourned to Philadelphia.

PUBLICATIONS.

THE publications of the Institute comprise:

1. The minutes of the Proceedings of each Meeting, published in pamphlet form.

2. Such of the papers presented or read by title at each Meeting as are furnished by the authors and approved by the Council for full publication. (In nearly all cases in which papers, the titles of which appear in the Proceedings, are not subsequently published, they have been withdrawn by the authors.) These papers are published separately in pamphlet form, and are marked "Subject to Revision." A small supply, beyond the edition distributed, is retained to meet subsequent demand. There are no copies on hand of papers read before 1880. The stock is nearly complete from 1880.

3. Annual volumes of *Transactions*, containing the list of officers, rules, etc.; the Proceedings; and the papers, *revised for final publication.* (In this revision after the preliminary publication, authors are permitted to use the largest liberty; and the changes and additions made in papers are sometimes important. It should be borne in mind, by those who study or quote a paper in the preliminary edition, that they may not have in that form the ultimate and deliberate expression of the author's views. It should be added, however, that in the majority of cases there is no essential change, the correction of typographical errors and additions of later information being the usual alterations.)

4. Special editions of separate papers, for which there is demand. These are fully revised, and are usually issued in pamphlet covers.

5. Books. (Under this head the only publications are an Index to Vols. I. to XV. inclusive, a Glossary of Mining and Metallurgical Terms, and a Memorial of Alexander Lyman Holley.)

All the foregoing publications are sent free to members and associates *not in arrears at the time of publication.* They are also for sale at the office of the Secretary, or are sent to purchasers by mail or express, charges paid, on receipt of the price by the Secretary, as follows:

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All communications and remittances should be addressed to R. W. Raymond, Secretary, P. O. Box 223, New York City.

RULES

ADOPTED MAY, 1873. AMENDED MAY, 1875, 1877, AND 1878, FEBRUARY, 1880, 1881, 1887, AND 1890.

I.

OBJECTS.

THE objects of the AMERICAN INSTITUTE OF MINING ENGINEERS are to promote the arts and sciences connected with the economical production of the useful minerals and metals, and the welfare of those employed in these industries, by means of meetings for social intercourse, and the reading and discussion of professional papers, and to circulate, by means of publications among its members and associates, the information thus obtained.

II.

MEMBERSHIP.

The Institute shall consist of Members, Honorary Members, and Associates. Members and Honorary Members shall be professional mining engineers, geologists, metallurgists, or chemists, or persons practically engaged in mining, metallurgy, or metallurgical engineering. Associates shall include all suitable persons desirous of being connected with the Institute, and duly elected as hereinafter provided. Each person desirous of becoming a member or associate shall be proposed by at least three members or associates, approved by the Council, and elected by ballot at a regular meeting (or by ballot at any time conducted through the mail, as the Council may prescribe) upon receiving three-fourths of the votes cast, and shall become a member or associate on the payment of his first dues. Each person proposed as an honorary member shall be recommended by at least ten members or associates, approved by the Council, and elected by ballot at a regular meeting (or by ballot at any time conducted through the mail, as the Council may prescribe) on receiving nine-tenths of the votes cast; *Provided*, that the number of honorary members shall not exceed twenty. The Council may at any time change the classification of a person elected as associate, so as to make him a member, or *vice versa*, subject to the approval of the Institute. All members and associates shall be equally entitled to the privileges of membership; *Provided*, that honorary members shall not be entitled to vote.

Any member or associate may be stricken from the list on recommendation of the Council, by the vote of three-fourths of the members and associates present at any annual meeting, due notice having been mailed in writing by the Secretary to the said member or associate.

III.

DUES.

The dues of members and associates shall be ten dollars, payable upon their election, and ten dollars per annum thereafter, payable in advance at the annual meeting. Honorary members shall not be liable to dues. Any member or associate not in arrears may become by the payment of one hundred dollars at one time a life-member or associate, and shall not be liable thereafter to annual dues. Any member or associate in arrears may, at the discretion of the Council, be deprived of the receipt of publications, or stricken from the list of members when in arrears for one year; *Provided*, that he may be restored to membership by the Council on payment of all arrears, or by re-election after an interval of three years.

IV.

OFFICERS.

The affairs of the Institute shall be managed by a Council, consisting of a President, six Vice-Presidents, nine Managers, a Secretary and a Treasurer, who shall be elected from among the members and associates of the Institute at the annual meetings, to hold office as follows:

The President, the Secretary, and the Treasurer for one year (and no person shall be eligible for immediate re-election as President who shall have held that office subsequent to the adoption of these rules, for two consecutive years), the Vice-Presidents for two years, and the Managers for three years; and no Vice-President or Manager shall be eligible for immediate re-election to the same office at the expiration of the term for which he was elected. At each annual meeting a President, three Vice-Presidents, three Managers, a Secretary and a Treasurer shall be elected, and the term of office shall continue until the adjournment of the meeting at which their successors are elected.

The duties of all officers shall be such as usually pertain to their offices, or may be delegated to them by the Council or the Institute; and the Council may in its discretion require bonds to be given by the Treasurer. At each annual meeting the Council shall make a report of proceedings to the Institute, together with a financial statement.

Vacancies in the Council may occur by death or resignation; or the Council may, by a vote of the majority of all its members, declare the place of any officer vacant, on his failure for one year, from inability or otherwise, to attend the Council meetings or perform the duties of his office. All vacancies shall be filled by the appointment of the Council, and any person so appointed shall hold office for the remainder of the term for which his predecessor was elected or appointed; *Provided*, that the said appointment shall not render him ineligible at the next annual meeting.

Five members of the Council shall constitute a quorum; but the Council may appoint an Executive Committee, or business may be transacted at a regularly called meeting of the Council, at which less than a quorum is present, subject to the approval of a majority of the Council, subsequently given in writing to the Secretary, and recorded by him with the minutes.

V.

ELECTIONS.

The annual election shall be conducted as follows: Nominations may be sent in writing to the Secretary, accompanied with the names of the proposers, at any time not less than thirty days before the annual meeting; and the Secretary shall, not less than two weeks before the said meeting, mail to every member or associate (except honorary members), a list of all the nominations for each office so received, together with a copy of this rule, and the names of the persons ineligible for election to each office; and if the Council, or a Committee thereof, appointed for the purpose, shall have recommended any nominations, such recommendation may also be sent to members and associates with the said list of all nominations made, but not upon the same paper. And each member or associate, qualified to vote, may vote, either by striking from or adding to the names of the said list, leaving names not exceeding in number the officers to be elected, or by preparing a new list, signing said altered or prepared ballot with his name, and either mailing it to the Secretary or presenting it in person at the annual meeting; *Provided*, that no member or associate in arrears since the last annual meeting shall be allowed to vote until the said arrears shall have been paid. The ballots shall be received and examined by three Scrutineers, appointed at the annual meeting by the presiding officer; and the persons who shall have received the greatest number of votes for the several offices shall be declared elected, and the Scrutineers shall so report to the presiding officer. The ballots shall be destroyed, and a list of the elected officers, certified by the Scrutineers, shall be preserved by the Secretary.

VI.

MEETINGS.

The annual meeting of the Institute shall take place on the third Tuesday of February, at which a report of the proceedings of the Institute and an abstract of the accounts shall be furnished by the Council. Two other regular meetings of the Institute shall be held in each year, at such times and places as the Council shall select, and notice of all meetings shall be given by mail, or otherwise, to all members and associates, at least twenty days in advance. Special meetings may be called whenever the Council sees fit; and the Secretary shall call a special meeting on a requisition signed by fifteen or more members. The notices for special meetings shall state the business to be transacted, and no other shall be entertained.

Every question which shall come before any meeting of the Institute, shall be decided, unless otherwise provided by these Rules, by the votes of a majority of the members then present. Any member or associate may introduce a stranger to any meeting; but the latter shall not take part in the proceedings without the consent of the meeting.

VII.

PAPERS.

The Council shall have power to decide on the propriety of communicating to the Institute any papers which may be received, and they shall be at liberty, when they think it desirable, to direct that any paper read before the Institute, shall be printed in the Transactions. Intimation, when practicable, shall be given, at each general meeting, of the subject of the paper or papers to be read, and of the questions for discussion at the next meeting. The reading of papers shall not be delayed beyond such hour as the presiding officer shall think proper; and the election of members or other business may be adjourned by the presiding officer, to permit the reading and discussion of papers.

The copyright of all papers communicated to, and accepted by, the Institute, shall be vested in it, unless otherwise agreed between the Council and the author. The author of each paper read before the Institute shall be entitled to twelve copies, if printed, for his own use, and shall have the right to order any number of copies at the cost of paper and printing, provided said copies are not intended for sale. The Institute is not, as a body, responsible for the statements of fact or opinion advanced in papers or discussions at its meetings, and it is understood that papers and discussions should not include matters relating to politics or purely to trade.

VIII.

AMENDMENTS.

These Rules may be amended at any annual meeting by a two-thirds vote of the members present; *Provided*, that written notice of the proposed amendment shall have been given at a previous meeting; *and Provided, also*, that the amendment or amendments so adopted shall be printed upon a ballot and sent, not later than the next distribution of printed matter, to all members and associates not in arrears for the preceding year (except honorary members and foreign members elected before February, 1880), and each person receiving the same shall be requested to return it to the Secretary with his written vote of Yes or No to each amendment, and his signature; and the President shall appoint as scrutineers three members or associates, who shall examine all of the said ballots which shall have been returned within one month from the date of their distribution, and shall report the result; and the Secretary shall publish and distribute to members, not later than the next distribution of printed matter, an announcement of the said result so reported, together with the text of the additional or amended rule or rules so adopted; and the amendment or amendments approved by the majority of the ballots so returned and reported shall become part of these rules from and after the publication of said announcement by the Secretary.

PROCEEDINGS.

DEDICATION OF THE HOLLEY MEMORIAL, NEW YORK, OCTOBER, 1890.

THE *Transactions* of the Institute (vols. x., 238; xi. 20, 222), and the Holley Memorial volume, issued in 1884, record the action taken by the Institute, and cordially shared by the American Society of Civil Engineers, the American Society of Mechanical Engineers, and by individual engineers throughout the world, in honor of Alexander Lyman Holley, a distinguished and beloved member, and, at one time or another, an officer of each of the societies named, whose death, January 29, 1882, was equally mourned by his colleagues and friends, at home and abroad. The numerous addresses, resolutions, and letters, contained in the Memorial volume published by this Institute, indicate, without exhausting, the evidences of the love and admiration with which he was regarded. Under resolutions of the three American engineering societies, the following joint committee was appointed to make arrangements for further suitable commemoration of Mr. Holley's character and career:

For the American Institute of Mining Engineers.—R. W. Raymond, Chester Griswold, John Fritz, Thomas Egleston, and G. W. Maynard.

For the American Society of Mechanical Engineers.—W. P. Trowbridge, J. C. Bayles, James A. Burden, Eckley B. Coxe, and R. W. Hunt.

For the American Society of Civil Engineers.—Charles Macdonald O. Chanute, W. G. Hamilton, I. Newton, and T. C. Clarke.

This committee was subsequently organized by the election of Mr. Raymond as chairman, and Mr. Macdonald as secretary and treasurer.

By request of the committee, a Memorial Address was delivered in New York, November 1, 1882, by R. W. Raymond.

The committee also raised, by a subscription, to which contributions were received from both sides of the Atlantic, a fund somewhat exceeding ten thousand dollars for a suitable memorial to be erected in New York. After many delays (the chief among which was due to the rule of the Park Commisson of New York city, preventing the erection in the public parks of any statue until a certain number of years has elapsed after the death of its subject), this memorial

was completed, and a most appropriate opportunity for its public dedication was offered by the meeting of the Iron and Steel Institute in the United States, in the autumn of 1890.

The memorial, which now stands in Washington Square, New York city, is shown in the accompanying frontispiece. It consists of a bronze portrait bust, of heroic size, executed by the eminent American sculptor, Mr. J. Q. A. Ward, and mounted upon a pedestal of stone designed by Messrs. Carrière and Hastings, architects, of New York City. The inscriptions are as follows:

HOLLEY.

Born in Lakeville, Conn., July 20, 1832.
Died in Brooklyn, N. Y., January 29, 1882.

In Honor of
Alexander Lyman Holley
Foremost among Those
Whose Genius and Energy
Established in America
And Improved Throughout the World
The Manufacture
Of Bessemer Steel
This Memorial is Erected
By Engineers
Of Two Hemispheres.

Mr. James Dredge, the distinguished editor of London *Engineering*, and an intimate friend of Mr. Holley, was invited by the committee to deliver the commemorative address in connection with the unveiling of the memorial, and it was arranged that the address should be given on Thursday afternoon, October 2, 1890, in Chickering Hall, and followed by the ceremony in Washington Square.

At the appointed time, Mr. Raymond, as chairman of the Holley Memorial Joint Committee, introduced to the audience gathered in the hall, and representing engineers of many countries, the orator of the occasion, who delivered the following address:

HOLLEY MEMORIAL ADDRESS.

By James Dredge, Honorary Member of the American Society of Mechanical Engineers.

Mr. President, Members of the Holley Memorial Joint Committee, Ladies and Gentlemen: Before commencing my address I propose, with your permission, to read to you two letters, which you will certainly listen to with interest and, I hope, with pleasure. The first is from Mr. James Forrest, the Secretary of the Institution of Civil Engineers, and a gentleman with whom many of you are acquainted.

The Institution of Civil Engineers,
July 31, 1890.

James Dredge, Esq.

Dear Sir: On learning that you have been intrusted with the privilege of unveiling in Central Park, New York, a monument to the memory of the late Alexander Lyman Holley, civil engineer, the Council of this Institution have considered in what manner expression could best be given on that occasion to the feeling of respect and esteem in which Mr. Holley was held by many members of this Council and of this Institution. As the result of their deliberations, I have been instructed to request you, in the name of the Council, to be so good as to make yourself the channel for communicating the sentiments of regard which the Institution entertains for his memory.

The late Mr. Holley was intimately known to several members of the existing Council, who have not only expressed their high appreciation of his exceptional skill and acquirements as a man of science and as an engineer, but also their warm affection for a man whose ability, sincerity, and purity of character appear to have endeared him, in no common manner, to those who were fortunate enough to know him well. Like testimony to the high professional and personal character of Mr. Holley has reached the Council from members of the Institution living in different parts of this country.

The details of Mr. Holley's labors, investigations, and services, as practically exemplified by his work, are well known to you and to the scientific world in general. I forbear, therefore, from entering upon them, although very much might readily be written on the subject. But what I must not abstain from mentioning, as I have been specially instructed to advert to it, is the eminent degree in which the late Mr. Holley's work, and his repeated visits to this country, assisted in bringing about, and in establishing, as the Council believe, upon an enduring basis, the cordial understanding which so happily exists between the great engineering professions of the United States and of the mother country. There is no member of this Council, and, I will venture to add, no member of this Institution,—large as it is, and widely as its members are spread throughout the world,—who does not greatly appreciate, and keenly enjoy, the friendly feeling which exists between the people of all classes in the United States and in this country. To attain such an end was ever the aim of the man with whose memory you are soon to be so honorably connected, and the Council trust that you will associate them and their fellow-members of this Institu-

tion with the forthcoming celebration by informing Mr. Holley's countrymen, if opportunity offer, of the esteem and honor in which his memory will always be held by them and by,

Dear Sir,
Yours truly,
JAMES FORREST,
Secretary.

The second letter is from a great man whose name is very familiar to you all; to whom you, in common with all the world, are indebted, and to whom you have already done honor. I refer to Sir Henry Bessemer, but for whom Holley's career, while it could not have been less brilliant, would have been largely modified. Sir Henry Bessemer writes:

DENMARK HILL, SURREY,
August 1, 1890.

DEAR MR. DREDGE:

The engineers of the United States having done you the honor to invite you to New York for the purpose of unveiling the statue of your old and esteemed friend, Alexander L. Holley, it has occurred to me that this occasion presents an opportunity of which, with your kind permission, I am most anxious to avail myself.

You are aware that the civil, mechanical, and mining engineers of America have invited the members of the Iron and Steel Institute to visit their country, and, with more than their usual generous hospitality, have provided a most interesting and extensive programme for their mental and physical enjoyment.

It is a source of very great regret and disappointment to me that nature has interposed an insuperable barrier between us by giving me a constitution that does not permit me to make the shortest sea voyage without absolute danger to life. But for this circumstance, the fact of my being in my seventy-eighth year would not have prevented me from accepting their generous hospitality, and seeing with my own eyes the enormous progress which the indomitable energy and perseverance of the citizens of the New World have made within the last twenty years.

Much as I should have felt interested and impressed with their great mechanical and engineering progress, I should have experienced still greater pleasure and pride in visiting their magnificent steel works and seeing the material that bears my name produced on a scale of greater magnitude than it is in any other country in the world. And when I reflect on the great honor conferred on me by this appreciation of my labors, I find it difficult to express the gratification it has afforded me.

In England, France, and Germany, many generous tributes have been accorded me by imperial personages, corporate bodies, and scientific institutions. From the nature of her Constitution, America has no orders of knighthood or other titles to bestow. But to a generous people a way may always be found of honoring those whom they desire to honor, and in giving the name of Bessemer to a rising city in the United States, they have created a living monument to my memory that will last for centuries after all the titles and medals I have received in Europe have passed away and been forgotten. It is in vain I look for words adequately to express my gratitude for this great memorial, but I gladly avail myself of this opportunity of thanking them, though but very feebly, for the honor done me, which far exceeds in generosity and value anything which my invention deserves.

It has been suggested to me that the Society of Mechanical Engineers of America would still further honor me by accepting my portrait. This I now send you, and I beg you to present it to them in my name, and at the same time to express the gratification its acceptance will afford me. Thanking you in advance for the trouble I am giving you,

I remain, dear Mr. Dredge,
Yours faithfully,
HENRY BESSEMER.

When I received from Dr. Raymond, in April last, an invitation from your Joint Committee to deliver the inaugural address that was to precede the ceremony of unveiling the Holley Memorial Statue, I was bewildered at the great and wholly unexpected honor thus suddenly pressed upon me. It was evident that the reason assigned in a letter, which I shall always preserve as one of my most valued possessions, was inadequate to account for the decision at which your committee had arrived. It seemed so clear that to an American alone belonged the privilege of delivering the oration in memory of one of America's greatest engineers, and certainly her most brilliant and best loved. And that you should cede to a foreigner this great privilege, and be content to accept cold and halting language when the bright and flowing eloquence of a hundred of your countrymen was at your disposal, remained for a long time a dark enigma to me. "We Americans have all said our say about him," so Dr. Raymond wrote; but who is there among you who would not tell once more with pleasure the oft-repeated story of Alexander Lyman Holley? It was not possible to accept this reason literally when there are so many of you who could hold an audience by their eloquence on a subject of which we never tire; who could recall memories the sorrow of which have lost their bitterness; who could do far more justice than I to one whose name is as freshly dear as it was on that dark winter day when he passed from us. Still less would I like to imagine that when you selected me for this high honor, you entertained any belief in my peculiar fitness, except that I am filled with a never-ceasing admiration and love for the man in whose name we are gathered together.

But though you preferred to cover with a light reason the real and noble motive by which you were directed to your resolve, this motive cannot fail to be known and appreciated throughout England,—throughout Europe,—where the memory of Holley is still kept green. To draw closer the bonds of brotherhood and of friendship; to show the world that while you claim the Master as your own, you wish to

pay a tribute to those beyond the sea who loved him too; to give fruition to one of the most earnest desires of his life, that of bringing together in close and friendly intercourse the engineers of the Anglo-Saxon race; these were the motives that I read concealed beneath the assurance that you had all "said your say" about Holley. Commenced last year, carried forward by leaps and bounds, through your self-abnegation in giving to England the post of honor to-day, you have made Holley's dream of union a reality, the brightness of which is dimmed only by the thought that he cannot know it is a dream no longer.

If I were able to do justice to the theme, I believe that there are few persons who have such good cause to speak with feeling and with eloquence upon it as myself. You will understand why, presently, if I succeed in making clear to you how deeply I am indebted to Holley, how great his influence for good has been upon my character and whole career. If I succeed in making you understand this, you will understand also why my admiration for him is so largely blended with reverence, and my affection with gratitude.

In framing this address, I have endeavored to carry out, as far as possible, the suggestion contained in Dr. Raymond's letter—"Now we want a voice from beyond the sea to speak for all those who appreciated his worth and work." To give the fullest effect to this desire I thought it only due to some of those who did most fully appreciate the worth and works of Holley, to let them speak for themselves, through me; and, as the representative of those whose voices I wish could have been heard here to-day, rather than as speaking for myself, I have dared to undertake the great responsibility that your generosity has laid upon me, and I trust you will believe that if my own individuality should be thrown into more prominence than might appear fitting, it is because my whole career has been strangely influenced and moulded upon those of Holley and the brilliant friend and companion of his early years, Zerah Colburn.

When first I passed the subject under general review, and, as it were, at a distance, it seemed to be pregnant with matter for discourse; but, as I approached it closer, much that was full of promise became intangible, just as delicate sentiments so often refuse to be crystallized in words. And when I read once more the noble monograph penned by Dr. Raymond, eight years ago, there seemed but little left for me to say; and that little, how impotently and feebly conceived, compared with the grief-inspired eloquence and the pains-

taking completeness that ennobled every line of that labor of bereaved friendship. The utmost I could hope to do was to throw here and there some side-light on Holley's story; to give you a better idea of the high esteem in which he was held in England than you have justly gathered from the measured, unimpassioned British phrases that concealed successfully our actual sentiments. But of Holley's history I can give you little that is new; possibly this is neither the time nor place to enlarge upon, or even to repeat what you have been already told so well. His was the story of all great men—a noble mission faithfully fulfilled; a life cheerfully yielded when his work was done. To him were entrusted, not five but fifty talents, and of all he could render an account manifold when he entered into his rest.

To my mind, the last eighteen months of Holley's life were the most beautiful of his whole career. He had passed through the valley of the shadow of death without a tremor; he had emerged from its cold shades with a keener sense of the emptiness of individual aims and ambitions, but with a higher appreciation of the true object of life—to keep always forward in the path of duty. His judgment and abilities had reached their culminating point, and his audacity —I can think of no better word to describe one of the greatest charms of his character—was somewhat tempered by a gentle melancholy, due, possibly, to a conviction that his days were not to be many. During that time, Holley conspicuously showed the fulfilment of all the rich promise of his youth; the powers that thirty years before had been latent, or were only beginning to burst forth, had attained their fullest growth, and all the lovable qualities of his nature had become mature.

In the preparation of this address two courses were open to me: to supplement Dr. Raymond's memoir, or to borrow from it. The first alternative had the disadvantage that I should present you with only a fragmentary and incomplete narrative; the latter, that you would be compelled to listen to facts with which you are already quite familiar. But, all things considered, it seemed to me that this was the best course to adopt. Some of you, possibly, have not read that memoir, its details may have faded from the memory of others, while those in whose recollection it is still fresh will not, I trust, find a brief summary wearisome.

The course of Alexander Holley's early years and education resembled those of most other well-born youths of the same period. He was the offspring of a union between the representatives of two

old New England families that had taken root and flourished for many generations in Connecticut, since they were transplanted from English soil. Deprived, almost from the day he saw the light, of a mother's care, he was fortunate enough to have that want supplied by the second marriage of his father; and the influence of the simple, rigid, and God-fearing domestic circle in which his earliest years were passed, nourished the noble impulses and high principles that were born in him, and strengthened them, so that they remained with him through life. He passed from the little district school to the local academy in Salisbury, thence to another in Farmington; from Farmington to Stockbridge, and from Stockbridge to Bridgeport, that he might prepare for Yale College. He did not commence his college career until 1850, when he was eighteen years old, so that his education, which had been carefully superintended by his father, was more than usually thorough and complete. Yale College did not have the honor of inscribing his name upon her books, because in 1850 the Brown University, at Providence, had founded a scientific chair, and it was determined that Holley should go there instead of to New Haven. After three years of college, Holley was well fitted to commence the battle of life. Twenty-one years of age, wondrously winning in manners and appearance, with a fine constitution, and indomitable energy, he was, indeed, well adapted physically for the struggle; and still more so intellectually. The strong natural bent of his character had been carefully strengthened and trained by himself from very early years. The influences of home were deeply rooted; his powerful memory, and almost limitless capacity for observation, had been developed by habitual and constant use. By nature he was an excellent draughtsman; and also by nature, he had the power of language; hence, from a very early age, he had a facile and rapid pencil; a keen and eloquent tongue. He was a leader in youthful debate; an acknowledged chief in college journalism. The boy was father to the man; and the engineering instinct strong within him had shown itself almost from the first dawn of his intelligence. Mechanics, the design of machinery, and especially everything that had to do with locomotive engineering, gradually took the first place in his mind, almost to the exclusion of collegiate subjects, and he made his start in the world in the shops of Messrs. Corliss and Nightingale, at Providence; there he served with credit as draughtsman, mechanic, engine-driver, and inventor; and it is a remarkable testimony to his great ability at that time that, without any special preliminary technical training, Messrs. Corliss & Night-

ingale parted from him regretfully, and at his own wish, because he preferred to devote himself to locomotive design and construction rather than to that of stationary engines. With only eighteen months' practical experience, Holley started out in search of employment in locomotive works. It was natural enough for him to meet with failure and disappointment; after about a month he fell into despair. "I am ready to sink;" he said; "for, if there is anything certain in this world, it is that I will never do anything but just this one thing, namely, build locomotives." Soon, fate was kind, and found employment for him in the locomotive works of Jersey City, where he remained long enough to gain valuable practical experience, and to convince all those about him of his great ability and energy. But his dream of always building locomotives for a living was to be a short one.

A young man, of his own age, and of brilliant genius, had, while Holley was receiving his education, raised himself from the menial life of a farmer's boy to that of the superintendent of the New Jersey Locomotive Works, and had quitted it before Holley had found the situation he was so anxiously seeking. Without education, this man had already come to the front as engineer and author; at fifteen years of age he had secured publication for a series of "Monthly Mechanical Tracts," which he issued at Lowell, in 1847. Humble as was his origin, his uncle had, years before his birth, brought the name he was to bear into notoriety, and the famous calculating boy, Zerah Colburn, was the object of wonder in scientific circles both in Europe and America. Beyond a natural gift of rapid calculation the uncle possessed no marked ability, whilst the nephew, less endowed as a mathematician, was, in other respects, probably the most marvellous member of the engineering profession of any country. A writer on mechanics at the age of fifteen, the superintendent of an important locomotive works at twenty—and this without any technical training except what could be gathered whilst he labored for his scanty living—Zerah Colburn at twenty-one commenced the career for which he was best adapted, and became editor of the mechanical department of the *American Railroad Journal*, in 1852. Of course he soon abandoned this position, and the following year undertook, as his own property, the *New York Railroad Gazette*. He was pushing the fortunes of this venture with that ferocious energy characteristic of the man, at the time when Holley obtained the situation he had been seeking; doubtless, the chief talk in the works was of the late superintendent's newspaper, and we may be

certain that Colburn was often at the locomotive shops getting information and seeking new subscribers. It seems but a poor little sheet now, that *New York Railroad Gazette*, for technical journalism has kept well to the front with the rapid and triumphant forward march of the engineer; but to me those flimsy pages are full of the deepest interest, from their being the first effort of a man who was certainly the ablest technical journalist we have ever seen. So it was through the means of this paper that Holley and Colburn first met, and for some years they were destined to tread together the upward path to fame.

I have said that the most boyish letters and essays of Holley showed gleams of literary talent which time and education kindled into a bright flame. In many respects, the tastes, aims, and ambitions of the two young men were similar. Holley had already realized the pleasure of seeing his work in print, and of being paid for that work; when he was eighteen years of age he had written quite a remarkable treatise on cutlery, which had appeared in *Poor's Railway Journal*, in 1850. It followed, as a matter of course, that he was as eager to write for Colburn's *Gazette* as Colburn was to obtain the assistance of such a congenial contributor. So, in a short time, the relations between the two drew closer, and Holley and Colburn entered into partnership, editing the little sheet conjointly. It was not long before a marked success attended their efforts; then Colburn wearied of the work, and in 1856 sold his share to Holley, and emigrated to Iowa, where he bought land with the proceeds of the sale, in obedience apparently to some passing fancy that prompted him for the moment to turn his hand to farming. For a year longer, the paper, under the changed name of *Holley's Railroad Advocate*, did fairly well; the new proprietor bore the business and editorial burden of the undertaking more systematically than Colburn, I am sure, but not with the same degree of feverish energy which Colburn always brought for a time into every new scheme he took in hand. In 1857, a great wave of commercial disaster swept over the country, and engulfed the *Railroad Advocate*. Colburn had wearied of Iowa, and returned to New York, where he and Holley, on the ruins of the paper, started another journalistic venture, which lived for two or three months as *Holley and Colburn's American Engineer*. Brief and unsatisfactory as these experiments in journalism had been, they nevertheless were attended with the most useful and important results to both men. They had lived by their pens for two years, which was no small matter; they had gained a vast amount of experience of a

varied character; they had, of necessity, come in contact with many engineers, and their writings had been sufficiently original and striking to attract much attention. The experience had, in fact, prepared them for their next venture, one which was destined to sever permanently the connection between the two men, and to open out for each new paths in widely diverging directions. The hard times of 1857 had absorbed what little money they possessed, but they could claim to be the owners of a more important capital—the confidence of many men of influence and means. It was notorious that at that time the construction and management of American railroads were in a deplorably backward condition compared with those of Europe, and the brilliant idea occurred to Holley and Colburn to get themselves made delegates of some of the principal railway companies in the United States; to travel over Europe, and to collect information on the railway practice of the Old World which should be made useful for the improvement and perfection of railroads at home. To secure the patronage and assistance of many influential persons was a work of no great difficulty; and in the summer of 1857 they commenced their first voyage across the Atlantic, a voyage which was destined to bear immediate fruit, and to lay the foundation for their brilliant and, alas! too brief careers.

Can you not picture to yourselves these two young men as they started upon their first and fateful voyage; full of generous enthusiasm; overflowing with life, and with the vigor that is born of intellectual power? Their early training was completed; their experience already wide; their powers of reception almost infinite. You can imagine that their high ambition, if it was vague, was as unbounded as they felt the power within them was of achieving it. How great their energy was, the result of the few weeks' visit to Europe on that occasion, still bears testimony, in the large volume they completed on the *Permanent Way and Coal-Burning Boilers of European Railways*. They were of an age when they could afford to burn the candle at both ends; and they did, under forced draught. In a memorandum by Zerah Colburn, now in my possession, I find the following reference to this book: "In 1857–58, I wrote the large work, entitled the *Permanent Way and Coal-Burning Boilers of European Railways*, Mr. Holley's name appearing as joint author in consideration of our sharing the cost of the undertaking, but more especially because of the assistance rendered by him in collecting information and preparing drawings." Those who can read between the lines of this memorandum will understand how large Holley's part was in the preparation of this book.

At that time, Mr. Daniel Kinnear Clark had attained a considerable reputation, both in England and America, by his very excellent book on *Railway Machinery*, and it followed, almost as a matter of course, that he was the first person in London sought out by Holley and Colburn when they arrived in 1857. At that time, I was a boy in Mr. Clark's office; I, like Holley, but in a very humble way, had been seeking despairingly for work, and a kind fate had placed me with Mr. Clark in the summer of 1857. I never can recall, without the deepest thankfulness, the mysterious working of Providence which, apparently by the merest accident, brought me in contact with these two men.

It happened that they, coming direct to London, had gone straight to the office of Mr. Clark, where I was alone, and thus I was the first person to shake hands with Holley on his arrival. Twenty-five years later, I was the last Englishman to shake him by the hand in London, when we said our final farewell. You cannot imagine the effect which the sudden apparition of the two young Americans in the gloomy London office had upon me. They appeared to me like beings from a superior world, so unlike were they to any persons I had ever met before. Even with my untutored and crude power of perception, I could feel that they were surrounded with an atmosphere of energy and intelligence; that they were overflowing with vitality. The one seemed to me a spirit of darkness, the other a spirit of light; and both so immeasurably my superiors, that I could do little more than gaze on them in wonder. Who could, at that time, have supposed that the influence of these two men was to be made, under Providence, to change and fashion my whole career; that they were to show me the path to a future infinitely brighter and higher than I could then have planned in my most sanguine dreams? Yet, so it was to be. Why they should, even then, have felt an interest in one so ignorant as myself, has always been a source of wonder to me. I suppose there must have been some mysterious and inexplicable bond of sympathy; I know that they exercised a strange influence over me; that, without my knowing it, they widened and strengthened my mind; that I absorbed much knowledge from them. When their brief visit came to an end, and they returned to New York, most of the light went out of my life, though their influence remained behind, especially that of Holley, whose bright individuality rested with me as an ideal, which I might, perchance, with time and constant effort, feebly imitate. The following year, Zerah Colburn returned to London alone; he became the editor

of the *Engineer*, and I, having sought him out, had the privilege of being closely and intimately associated with him until the curtain fell upon his tragic end. As no doubt you know, after he returned to England, in 1858, he made his permanent home there, with the exception of a few months in 1860, which he passed in Philadelphia. You are also aware that it was he who founded the London *Engineering*, in 1866; into the management of this journal he threw for the first three years the whole force of his great erratic powers. After their first visit to London, and their joint publication of the volume on European railways, Holley and Colburn, without becoming actually estranged, had but little in common. The dark and fiery genius of the one was, in fact, so opposed to the trained talents and noble soul of the other, that it was impossible for any real and lasting sympathy to exist between them.

With Holley's return to America in 1858, or shortly afterwards, commenced the journalistic epoch of his career. For some years he was a valuable and industrious contributor to the New York *Times*, and certainly never before or since has any daily newspaper been so fortunate in the services of an engineering contributor. In 1859, and again in 1860, he visited England chiefly to write upon a subject which was then attracting the attention of the civilized world, the Great Eastern. This business brought him into intimate contact with the engineers of that vessel, Brunel and Scott Russell. With the last named the acquaintance ripened into a warm personal friendship which always remained. The New York *Times*, the *Railway Review*, inventions connected with the steam-engine and permanent way, and miscellaneous literary employment kept Holley closely occupied and brought him plenty of reputation, though but little money, until the War of Secession. It is needless to say that Holley, like a good patriot, offered his able services to the government, and it is interesting to add that his offers were uniformly declined. It was during this great national crisis that Holley, while still holding on to his main support,—a brilliant and facile pen,—had the opportunity of taking up work more congenial to him. He was employed by Mr. Stevens in 1862 to visit Europe in order to obtain information that might be useful in the construction of the Stevens battery, on which so much time and money, as well as, skill, were wasted. Mr. Scott Russell, who at that time had considerable interest in naval and military circles, rendered Holley good service in obtaining for him introductions and information, while his own presence of mind and coolness often stood him in good stead in obtaining access to places the official doors of which were closed against him. It was

this new connection that turned his mind to the production of that important hand-book on *Ordnance and Armor*, which, when completed, was destined to remain for so many years the standard book of reference on the subject. Like every piece of work to which Holley set his hand, this treatise bears on every page evidence of his capacity for taking infinite pains. The objection has been made by friendly critics in this country that *Ordnance and Armor* was scarcely worthy of its author. I do not agree with this criticism; and in this I am sustained by one of the ablest practical designers of artillery in the world, Mr. G. Canet, who has written to me on this subject as follows:

You have told me that recently you have had frequent occasion to study the *Ordnance and Armor* of A. L. Holley. You could not refer to a better authority; for it is at once the most complete and the most conscientious work on the history of artillery that has ever been published. The name of Holley recalls to me many souvenirs of the commencement of my career, and I want to tell you that for me no book on the same subject has ever been of so much value. Holley wrote it at a time before artillery had become a science, properly so-called; when even the phenomena attending the combustion of gunpowder were very imperfectly understood. At that time the strains to which gun-carriages were subjected were not accurately known, and correct gun design was impossible, because the pressures exerted upon the bore of a cannon could not be measured. When Holley's book appeared there existed only a few theoretical essays, the records of some doubtful experiments, the ill-sustained claims of a few manufacturers, and these were all scattered through different memoirs and reports. Holley was the first to make a systematic summary of all these documents; to pass in able review material, modes of construction, ballistical data, firing experiments, and penetration of plates. This last-named portion of his work had an especial value, for when he wrote the art of constructing ironclad ships was in its infancy. What I find most remarkable in this work of Holley is, that writing at the period from which modern artillery may be said to date, he made all the data he could obtain the object of minute and careful study; that every line bears the stamp of his own work; that it is an earnest study, and in no sense a piece of bookmaking. Every page contains abundance of detail, accuracy in facts, and a sureness of judgment which is truly admirable. His considerations on the resistance and elasticity of material, and on the effects of strains and vibrations, show the large grasp of a truly scientific mind. His investigations on the subject of reinforcing rings had at the time a high value, and to-day are of considerable interest. He showed conclusively that he understood the absolute importance of gaining information at every step by experiment, and that theory is but of comparatively little value in gun manufacture. It is difficult to realize all the changes that have taken place since his time in the construction of ordnance. Steel of high quality scarcely existed, and an entire revolution has taken place in design and in construction. We who have followed carefully all the transformations that have occurred in the art during twenty years, and who were familiar with it when Holley wrote, gladly do homage to the man who could produce such a book from such materials. For my own part, I owe him a debt of gratitude, for it was from his pages that I studied most often, and whence I drew my first clear notions of artillery. I reverence him not only as a savant, but as a master.

G. CANET.

I thought it would give you pleasure to hear the independent testimony of an eminent Frenchman who never knew Holley except through his writings on this special subject. When this book was published, the art of heavy gun construction was far more tentative and experimental than it is now, and the classified collection of all information available up to the date of its publication served as a great aid in advancing a science then in its infancy. The strictly literary period of Holley's career was now drawing to a close, and though he could never, as long as he lived, abandon his innate love for writing, he was no longer compelled to rely on his pen as a means to live. In May, 1863, he finally decided upon turning his talent and energies into another channel, one that was destined to lead him to fame and moderate prosperity.

The practical value of the Bessemer process for manufacturing steel had by this time become widely recognized; and it followed almost as a matter of course that Holley, who was always on the lookout for some new and good thing, felt intuitively that the time for which he had so long waited had at last arrived. The story of the Bessemer process was at that time no new one. Seven years before, at the Cheltenham meeting of the British Association for the Advancement of Science, Mr. Bessemer had made the first public announcement of his invention. The title of the paper excited derision,* its reading wild enthusiasm. Mr. James Nasmyth said of the samples shown to the astonished audience: "Here are true British nuggets." A wave of excitement passed over the country. One result of this paper was that almost immediately a sum of $250,000 was offered for the English patents; another, that within a month £28,000 had been paid for licenses to work the process in Great Britain alone. Then swiftly followed a fierce reaction, consequent on failures due to hasty experiments and crude trials. Abuse of the inventor was as hearty as the praise had been fulsome a few weeks before.

The newspapers of the day were unanimous in condemning the process as delusive and useless, and referred to it as "a brilliant meteor that had flitted across the metallurgical horizon, dazzling all beholders for a moment, only to die out and leave no trace behind." As a lasting tribute to the sagacity of the Council of the British Association, I may add that that astute body refused to admit the paper

* "On the Manufacture of Malleable Iron and Steel without Fuel," read August 13, 1856.

in their published transactions. In the face of terrible difficulties and discouragements, Mr. Bessemer worked steadily on, and in the course of a few years he established works at Sheffield, where the process was still in an experimental stage when Holley first visited them. In a letter I recently received from Mr. W. D. Allen, the brother-in-law of Sir Henry Bessemer, and who was then, as he still is, the manager of those works, the following appreciative remarks about Holley occur:

BESSEMER STEEL WORKS, SHEFFIELD.
July 14, 1890.

MY DEAR MR. DREDGE:

I have indeed many cherished recollections of my relations with Mr. Holley, and if the few following facts have any interest, pray make what use of them you think proper.

Mr. Holley came to Sheffield in 1864. He was then a bright, intelligent young fellow (he did not look one-and-twenty), genial and open-hearted to a fault. America was never represented by a more perfect gentleman, or one more calculated to win the respect and esteem of all those he came in contact with in this country. Indeed, the handsome young American not only won our esteem but our hearts. He came with the intention of gathering what knowledge and facts he could relative to the Bessemer process of steelmaking, and with that view he passed several months at our works, observing and mastering the details of the process, so that I saw a good deal of him, and was associated with him during the greater part of his visit.

He applied himself very assiduously to the object he had in view; was mostly at the works by nine in the morning, and frequently remained till nine or ten at night to witness some special operation; for I must tell you that at that time our own career was so young that we were frequeutly occupied in the application of our material to some new and useful purpose. In these experiments Mr. Holley always took much interest.

As regards the process itself, he was most enthusiastic, and from the first pronounced it a "grand thing for America." His desires and aspirations were concentrated upon introducing it into that country, and his efforts were mainly instrumental in doing so.

It gave me very great pleasure when I heard that his countrymen had decided that the memory of so distinguished a man and so great a benefactor to his country should not be overlooked.

I am, dear sir,
Yours very truly,
W. D. ALLEN.

The Sheffield Bessemer Works, in 1864, were on a very small scale; they only comprised two 3-ton converters and a corresponding plant. Holley came from America upon this business as the representative of Messrs. Corning, Winslow & Co., a firm that was afterwards changed to Winslow, Griswold & Holley. His mission was to purchase the Bessemer patents for the United States, to learn the

process, and afterwards to inaugurate the new industry at home. As he said, that journey was no pleasure trip; but he came back successful on all points, and the following year the works at Troy were built.

As is the case with so many great inventions, a number of active minds were turned towards the manufacture of steel, stimulated thereto by Bessemer's success and his reputed failures. Prominent among these in the United States was W. Kelley, of Eddyville, Kentucky; and your talented countryman, Mr. W. F. Durfee, addressed himself to the practical development of this invention; it was he who constructed the experimental steel works at Wyandotte, Michigan. This was in 1862, the year of the great International Exhibition in London, where Holley had ample opportunity of studying the products of the Bessemer converter. We may be certain that what he saw there convinced him of the importance of the process for the United States. His judgment was confirmed by that of Zerah Colburn, who, with his usual foresight, had some time before become a strong advocate of the process. Writing in the London *Engineer* in 1862, Zerah Colburn said: "The fact, that as a substitute for wrought iron, at least where facility of welding is not of special consequence, the Bessemer metal is now known to be of established trustworthy character, is of itself a most important one; and as those who fully know the process are aware that ingots can be turned out by it at £6 per ton, and finished rails at less than £10, we have only to await the time when through competition this metal has been cheapened to an extent which will insure its general application for the many purposes to which it is so admirably fitted."

When Holley returned to America early in 1864, having purchased the American patents and learned the process, he and his associates found themselves confronted with the risks of costly litigation. It was claimed that the Kelley patents preceded and covered those of Bessemer, but after some delay this difficulty was removed by compromise. Indeed, on Mr. Durfee's own showing, the costly experiment to prove the value of the Kelley patents was really only a test of the Bessemer process. In an interesting paper read before your Society of Mechanical Engineers in 1884, Mr. Durfee says, with perfect frankness: "I possessed myself of all the information attainable concerning the Bessemer process, and as it was confidently expected that Mr. Z. S. Durfee would be successful in his efforts to purchase the American patents issued to Bessemer for the various forms of apparatus for the production of steel by the pneumatic method, it was thought only to be anticipating the acquisition of

property rights, to use such parts of his invention as seemed suited for the purpose." The Wyandotte experiment developed no great results, but for it can be claimed the first production of Bessemer steel in the United States, and the first output of steel rails. After the return of Holley to America in 1864, the Troy Works were started, and the first charge of Bessemer metal was made on February 16, 1865. Let me refer you to Mr. R. W. Hunt's interesting paper on the original Bessemer steel plant at Troy, read before your Society of Mechanical Engineers in 1884. You will see from it that Troy was only a small works with two 2-ton converters, but such capital steel was produced, that enterprise and capital were convinced. During the month of May, 1865, less than twelve weeks after the works were started, eighty charges were made, giving 118 tons of steel with 16.7 per cent. of loss; immediately after this the works were extended by the installation of a plant that included two 5-ton converters; Mr. Z. S. Durfee assumed the charge of the establishment, and Holley proceeded to construct steel works in other parts of the country, commencing with those of the Pennsylvania Steel Company at Harrisburg.

The thorough efficiency of the Bessemer process having been thus demonstrated, there was no limit to the supply of capital required for the erection of steel works, and Holley rapidly found himself almost unable to satisfy the demands made upon him for designing and starting new establishments. He, however, never knew what it was to have too much occupation; and the finest steel works in your country bear evidence of his skill and industry: those of North Chicago and Joliet, the Edgar Thomson Works at Pittsburgh, the Vulcan Works at St. Louis, the Cambria, the Bethlehem, the Scranton, and other works, were built under his instruction and from his plans. But more than this; Holley completely modified the details of the Bessemer process as he had learnt it in England; he adapted it to the special requirements of the American manufacturer, who was enabled, through the modifications he designed, to produce those stupendous outputs of steel, the records of which were so long received in England with incredulity. This is what the joint inventor of the Basic process, Mr. Percy C. Gilchrist, wrote to me upon the subject of Holley's modifications and improvements:

101 Palace Chambers, Westminster, London,
July 14, 1890.

I have the greatest admiration for the work Holley achieved, as I think that the present proud position of the American steel trade is very largely due to his genius. A lesser man would have copied the European steel works, but instead of

this he picked out the weak places, and in the American works that were fortunately subject to his advice, one sees how their defects were avoided by him, how break-downs were eliminated, labor saved and output consequently increased.

Holley, from the very first, appreciated the Basic process, and was most kind and generous in his appreciation; he suggested at an early date that the more rapid wear of the Basic lining of the converter should be met by removing the shell of the converter with its worn-out lining from its belt, and by replacing it with a fresh shell ready-lined; this plan was put into practice by Mr. Arthur Cooper at the North-Eastern Street Company's Works at Middleborough, and is giving excellent results.

I regret to say that I saw very little of Holley myself, but my cousin, the late Sydney Gilchrist Thomas, saw a good deal of him; they had much in common, both in ability, go-aheadness and perseverance, and they had, I know, an intense admiration for each other.

Yours very truly,

PERCY C. GILCHRIST.

One of your own engineers, Mr. R. W. Hunt, has summarized Holley's work in this direction. He says:

"The result of his thought gave us the present accepted type of American Bessemer plant; he did away with the English deep pit, and raised the vessels so as to get working space under them on the ground floor; he substituted top-supported hydraulic cranes for the more expensive counterweighted English ones; he put three ingot cranes round the pit instead of two, and thereby obtained greater area of power; he changed the location of the vessel as related to the pit and melting-house; he modified the ladle crane and worked all the cranes and vessels from a single point; he substituted cupolas for reverberatory furnaces; and last, but by no means least, he introduced the intermediate or accumulating ladle, which is placed on scales, and thus insures accuracy of operation by rendering possible the weighing of each charge of melted iron before pouring it into the converter. After building such a plant, he began to meet the difficulties of details in manufacture, among the most serious of which was the short duration of the vessel-bottoms, and the time required to cool off the vessels to a point at which it was possible for workmen to enter and make new bottoms. After many experiments, the result was the Holley vessel-bottom, which either in its form as patented, or in a modification of it as now used in all American works, has rendered possible, as much as any other one thing, the present immense production."

Vastly as all these modifications tended to increase the output, I think it is necessary to include one other factor that has helped to obtain such extraordinary results, the perfect organization of the managing engineers and the remarkable efficiency of the American workmen. On this point a prominent English steel-maker wrote to me recently, after a visit he had paid to several of your leading works: "Of course, the output was very large from each pit, and this was attained by great facilities for bringing up and getting away

materials, and keeping everything at high pressure. The men, I dare say, are paid well, but it was hot weather when I was there, and they were certainly selling their lives."

But apart from organization and skilled labor, the marvellous growth of production from Bessemer vessels in the United States must be credited to Holley. The fifteen years' development may be briefly summed up as follows: In 1865, the output from two 2-ton converters in a month was 118 tons; in 1880 the output from two 5-ton converters was 14,000 tons. In 1866 the Pennsylvania Railway Company purchased from Messrs. John Brown & Co. 5000 tons of Bessemer rails at $120 (gold) per ton; last year the production of Bessemer rails in the United States was about 1,644,000 tons. In the year 1868 the total output of Bessemer steel in America was 8500 tons; the same year in England it was 110,000 tons. In 1879 your production equalled ours, and from that date has always exceeded it, until the last year it reached the amazing figure of 3,169,643 tons, which was one million tons more than that of the United Kingdom. Since the first charge was blown at the Troy Works in 1865, there have been produced in America upwards of 25 million of tons of Bessemer steel. And this is the record of the last eighteen years of Holley's life.

So the busy years passed, bringing with them a rich, if tardy, harvest of fame to Holley. He continued to be recognized as the one great authority in America on Bessemer steel; in Europe his reputation stood nearly, if not quite, as high. We all remember the prominent part he took in the Centennial Exhibition of 1876; his admirable reports on the iron and steel exhibits of that year, which were followed by a long and masterly series of articles on the steel works of the United States—a series which, unfortunately, remained unfinished. Some of us have had the privilege of studying his annual and confidential reports on the steel manufacture of the world. These reports are marvels of technical skill and labor; they attest also the unlimited confidence which he commanded both at home and abroad; in the steel works of Europe, as well as those of the United States. They were the result of periodical visits to England and the continent, and were, during the last years of his life, prepared solely for the use of the Bessemer Association, with which he had entered into confidential relations. The gates of every steel works in Europe were open to him, and from him no manufacturers' secrets were withheld. It was quite a unique position which he then occupied; one for which his long training, high intelligence, and

perfect integrity absolutely fitted him. Manufacturers knew well that their secrets were safe in his hands; that if they gave him much, he brought still more to them; that his visits were always useful and suggestive, and that the exchange of American experiences for their own never left Holley their debtor. There was also another aspect to these annual visits. Time, hard work, and the cares of a busy life, had never withered the great charm of Holley's character; but, on the contrary, his personal influence and indefinable power of attraction had increased with years. He was loved as much as he was admired; he was as popular in France, in Germany, and in Belgium, as he was in England or the United States. I could have laid before you to-day proofs of this till you would have wearied of them. You have already heard how the council of an institution, respected throughout the world—the Institution of Civil Engineers —have expressed themselves about Holley, speaking from personal knowledge of him. You have heard what were the sentiments of Mr. Allen, in the early days of the Bessemer industry, sentiments which grew warmer with the passing years. Let me read you a few more expressions of opinion from distinguished men in England and the continent of Europe. Mr. James Riley, of the Steel Works of Scotland, Glasgow, writes:

I regret that it will not be possible for me to be present on the interesting occasion when the monument to Mr. Holley will be unveiled and you will deliver the inaugural address, and that I shall therefore be deprived of the privilege of thus manifesting my sympathy with the sentiment which will culminate in the ceremony of that day.

It was my privilege and pleasure to know Mr. Holley during several years, in the latter of which I think I may say that our acquaintance became as intimate as the circumstances of time and distance rendered possible. I think I am correct in stating that for the last ten years of his life he never came to Europe without visiting me, the last occasion being the one he left us bravely struggling with the disease to which he finally succumbed.

During these years of acquaintance of more or less intimate character, I had learnt to admire and esteem Mr. Holley, and the more I knew of him the greater was the regard, and may I say affection, I felt for him. One admired his great abilities, esteemed him for his frank, honest, manly nature, but, more than this, one was drawn to the man by the grace and charm of manner and conduct which he possessed in such high degree. Feeling thus regarding Mr. Holley, I am glad to know that the engineers of America have taken steps to perpetuate his memory, that we are invited to participate in this action on their part, and that you are to give expression to our admiration and esteem on the occasion of the inaugural ceremony.

Yours faithfully,

JAMES RILEY.

Mr. E. Windsor Richards writes to me as follows:

LOW MOOR IRON WORKS, NEAR BRADFORD, YORKSHIRE,
August 11, 1890.

JAMES DREDGE, ESQ.

Dear Sir: I am very much pleased to learn that a statue is to be erected to the memory of our lamented friend, A. L. Holley, in Central Park, New York, and will be unveiled by you. It would be very gratifying to me to be present on that occasion, as I held him in the very highest esteem. It was sad that he should have gone from us when in the full ripeness of his career, when his abilities, sound common sense, and good qualities had become so well known to, and appreciated by, the leading metallurgists and engineers in the old country.

Holley had a very difficult part to perform, one requiring great judgment and knowledge of men. Few succeed as he did. Manufacturers generally most jealously guard their trade matters. Holley disarmed them by being always as ready to communicate information as to receive it; he had always something of the highest interest in engineering and metallurgical matters to discuss.

Nine years ago at this season he was directing the route for my visit to the States, a visit which afforded me so much pleasure and instruction, and which added many more names to my list of American friends.

I am, indeed, glad that so many of his countrymen have joined to do honor to the memory of a man so eminent, so amiable, and so genial.

Yours faithfully,

E. WINDSOR RICHARDS.

LONDON.

From France I bring you a message from Messrs. Schneider et Cie, of Le Creusot:

LE CREUSOT, July 18, 1890.

SIR:

We have the honor to acknowledge the receipt of your letter of the 14th inst., to which we hasten to reply. It was a fortunate inspiration to combine, as you have undertaken to do, the expressions of opinion of those engineers who had the privilege to hold relations with Alexander L. Holley, and we are grateful to you for having thought of us, who have always preserved such an agreeable recollection of our cordial intercourse with him in the past.

The bright intelligence which he brought to bear on all metallurgical questions; the great range of his information which an indefatigable energy enabled him constantly to extend; the readiness and grace with which he lent himself to the exchange of ideas, which were always valuable coming from a man of his high intellectual standard, secured for him the highest rank in the esteem of those who knew him, and we are happy in having the privilege to offer to his memory, with our fullest sympathy with the engineers of the United States, the sincere testimony of our esteem and of our highest consideration.

SCHNEIDER ET CIE.

From Germany, the Ruhrort Steel Company write:

RHEINISCHE STAHLWERKE IN RUHRORT,
July 14, 1890.

DEAR SIR:

In reply to your esteemed favor of 7th inst., we beg to express to you our never-ceasing regret at the loss the profession sustained in the death of A. L. Holley, and we are deeply gratified to know that his countrymen are about to commemorate his important services to industry and science, and to keep alive his memory by the erection of a monument to him.

We take advantage of the fact that you are going to New York on the occasion of the inaugural ceremony, and we hereby request you to impress upon our American friends and colleagues our complete and heartfelt sympathy with them in the realization of their honored work. The great esteem in which we held our late friend and brother engineer, fills us with a sentiment of keen regret that we shall not be able to assist personally at the ceremony. We thank you, therefore, for having undertaken the kindly mission of acting as the interpreter of our high regard for the memory of our dead friend, for whom we shall always retain the warmest affection.

With our sincere regards, we are, etc.,

THE DIRECTORATE OF THE RUHRORT STEEL WORKS.

From Austria, I have the following communication:

WITKOWITZ STEEL WORKS, AUSTRIA,
July, 1890.

DEAR SIR:

It was with heartfelt satisfaction that we learned from your esteemed favor of the 7th inst. that the memory of that great American engineer, A. L. Holley, is to be honored and preserved by a memorial column, and that this monument is to be unveiled in the presence of those European engineers who are invited to enjoy the hospitality of our American colleagues. We owe Mr. Holley much for the valuable information and suggestions we received from him when he visited our works, and his memory will always be warmly cherished by us, as much on account of his rare amiability and charm of character as for his great professional talent. We ask you to extend in our name and on our behalf a very warm and hearty greeting to the engineers of the United States, whose visits to our works have always been, and ever will be, welcome.

We are, dear Sir, yours faithfully,

THE DIRECTORS OF THE WITKOWITZ STEEL WORKS.

From the John Cockerill Works at Seraing, M. Greiner, an able metallurgical engineer, and an old and valued friend of Holley, intrusts me with the following message to you:

SOCIÉTÉ COCKERILL, SERAING, BELGIUM,
August 6, 1890.

DEAR MR. DREDGE:

At the moment when the engineers of the United States are preparing to render solemn homage to their illustrious *confrère*, A. L. Holley, you have done me the

honor to ask me to communicate through you my opinion of this eminent engineer who was so prematurely removed from science, from industry, and from the affection of his many friends. I cannot better reply to your kind invitation than by recalling some personal souvenirs that are associated with the numerous visits made by Holley on the Continent after he had quitted the Troy Works, where he made his *début* in Bessemer steel-making manufacture in America. This establishment had many points in common with the steel works of Seraing, of which I was the engineer as long ago as 1864, and this was the cause of our friendship, as well as the reason of our first meeting. From the commencement, Troy and Seraing manufactured steel by the Bessemer process, of many classes and for all purposes. Holley thought as I did,—and confirmed his opinions by the results he obtained,—that the converter, properly employed, could be made to yield all the qualities of steel required by commerce, from the more ordinary metal adapted for rails to the high-class steels required for ordnance, rifles, springs, etc. When, extending the field of his operations, he was called successively to assist at the creation of all the steel works in the United States, our relations became more intimate,—not because he had much to learn from us so far as mechanical applications or general designs were concerned, because in all these things he was a past master, but because he was struck with the extreme moderation in our cost of manufacture, and it was in the comparison of Belgian with American net prices that he found matter for study and reflection which strengthened our relations and made them more closely intimate. And then there was something else which drew us to him. In the midst of his incessant journeyings from works to works across the continent; in his indefatigable researches after new details and processes, or fresh applications; with that untired ardor which astonished us all, he loved to come to us sometimes to rest for a few days, intervals, alas! that occurred too seldom and were always too short, and he found in the beautiful country that we loved to traverse together, always discussing technical subjects, landscapes that recalled to him, on a small scale, your beautiful Pennsylvania.

Besides the engineer and the man of science, Holley was a true artist, with elevated sentiments and impressionable feelings before the beautiful spectacles of nature; and besides this, he was also a man who did not disdain to mingle bright pleasantry with the gentle philosophy with which he knew how to surround everything. The first time that I saw Holley we visited together the works at Terre Noire to examine the results obtained by the addition of a certain quantity of phosphorus in rail steel without detriment, so it appeared, to their quality. The last time we met it was still the question of phosphorus that brought us together, but this time on a more serious matter. We visited, with Thomas (alas! he is dead, too) and Mr. C. P. Sandberg, the beautiful grotto of Han, on the Lesse. In one of these subterranean halls, strangely and ruggedly broken up by stalactites and stalagmites that our guide illumined for us with the glare of his torch, Holley said laughingly to Thomas that he felt as if he were in the interior of a gigantic basic converter. Now, this was the time (1878) when poor Thomas had not succeeded in making a satisfactory lining to his converters, and which, after every charge, presented the most curious and wild-looking appearance. It was a gift of Holley's that he was able, on the spur of the moment, to make the most striking and humorous comparisons; and in his lighter moments, in the company of his friends, his conversation scintillated with the bright sparks of his wit.

I do not feel, my dear Mr. Dredge, that I have any right to say all that I think about Holley's noble qualities. Voices with more authority than mine—and yours especially—will bear witness to the splendid work of the engineer, the remarkable

publications of the man of letters, the undisputed judgment of the man of science; but if to this testimony I have nothing to add, I cannot refrain, at the risk of repeating that which doubtless you will feel yourself called upon to say, that I can never hope to meet in the course of my career a man whose soul was more noble, whose character was more loyal and more frank, whose tact and sentiment were more delicate, and whose charming manner, whether in business or in private relations, was more striking. He was, if I may thus dare to express myself, a true type of the "gentleman engineer," and it is thus that he will ever remain in my memory. It is with very deep regret that I have to abandan my intention of joining the members of the Iron and Steel Institute in the visit to their *confrères* of the United States, who have chosen the happiest moment to immortalize the name of their famous compatriot Alexander Holley, by unveiling his statue in the midst of the sympathetic acclamations of the engineers of the Old and New Worlds.

Believe me, dear Mr. Dredge,

Yours most faithfully,

A. GREINER.

Had it been necessary I could have multiplied these evidences of friendship and good will tenfold. But I think I have brought you sufficient "voices from beyond the sea to speak to you of Holley's work and worth;" enough evidence of how he was admired and esteemed abroad."

A passing reference should be made to the Thomas-Gilchrist process of making Bessemer steel, which was brought into prominent notice only three years before the close of Holley's career, at a time when his continued efforts had already broken down his magnificent constitution. The ill-fated Mr. Thomas in some respects resembled Holley, and they had sufficient subjects of interest in common to create a strong bond of sympathy between them. Always eager to know of what was new and of apparent value, Holley speedily made himself master of the basic process and passed considerable time with the inventors at the works where the method was in operation. He quickly formed a high estimate of the value of the process, and made preliminary arrangements with Mr. Thomas for its introduction into the United States. Partly as a consequence of this arrangement, Thomas visited the States in 1879 and passed some weeks with Holley, inspecting the principal of your steel works. It was on one of these occasions that Holley made the remark to Thomas, when he was fatigued and wanted to sit down, "that he must send over to England if he wished to find a cool ingot-mould." I think that Holley's early death was the sole reason why the Thomas-Gilchrist process is so little employed in the United States; the good opinion that Holley formed at the commencement has been fully justified by its large and successful adoption on the continent of Europe. In

this connection, Mr. Arthur Cooper, manager of the North-Eastern Steel Company, of Middlesbrough, and one of Holley's friends, writes me as follows:

THE NORTH-EASTERN STEEL COMPANY, LIMITED.
Middlesbrough, August 4, 1890.

MY DEAR SIR:

As one of Holley's great admirers, it affords me pleasure in bearing testimony of the very high degree in which he was appreciated by me. It fell to my good fortune to make his acquaintance in Sheffield, early in the seventies, whilst he was paying one of his many visits to this country, and this acquaintance soon ripened into a firm friendship, for although Holley's visits at that time were undertaken chiefly for the purpose of posting himself as to the means in use here of carrying out in detail the acid, Bessemer, and Siemens processes, so great were his abilities, so bright his intellect, and such was his genial temperament, that in imparting to him any information, it always seemed to me that the benefits received were greater far than the benefits conferred, because in discussing the various points Holley had always some suggestion to offer well worthy of the most careful consideration, if not, indeed, of immediate adoption; and afterwards, when the basic process was introduced, and he became intimately associated with the late Mr. Thomas, he was amongst the first to estimate at their true value the mechanical difficulties that had to be encountered, and his genius promptly devised a ready means of overcoming the greatest of these difficulties. Hence it happens that there are few steel works in this country or on the continent that have not profited in a greater or less degree from his brilliant talents. With such tributes to his memory it is scarcely necessary to say that he was regarded by me as one of the cleverest and most reliable of steel-works engineers, whose death was a great loss, not only to his country, but to the world.

With regard to Holley's private character, truthfulness, honesty of purpose, generosity and sincerity, were the essence of his life, and no one who met him casually could help admiring him as a man and a true gentleman, whilst his more intimate acquaintances will ever cherish the recollection that they once had the privilege of knowing such a man and of calling him friend.

I am, my dear sir, yours very truly,

JAMES DREDGE, Esq. ARTHUR COOPER.

Did time permit I should like to dwell at much greater length on the brilliant and successful years of Holley's career; the period of an abundant harvest, the seed of which had been sown with toil and judgment so many years before. But I must pass on to review very briefly some of the minor though important incidents that complete the picture of his busy life. The professional honors that were so fully his due, fell thickly enough on him during the later years. He was elected President of your Institute of Mining Engineers in 1875; he was Vice-President of your Society of Civil Engineers in 1876; he founded your Society of Mechanical Engineers, of which

Society you did me the high honor to elect me an honorary member; in 1877 he became a member of the Institution of Civil Engineers, a body which we in England are proud to regard as the parent of all the Anglo-Saxon engineering societies. In connection with this election, I think you will be interested to hear the summary of Holley's life and works as told by himself in detailing his qualifications for membership.

"In 1853 he graduated in the scientific department of Brown University; was then for one and a half years under Mr. E. H. Corliss, locomotive engine builder, and one year under Mr. W. G. Hamilton in the New York Locomotive Works; from 1856 to 1862 he was engaged partly in the literary branch of the profession, and partly as an assistant engineer in the locomotive department of the various railways, and also under Mr. E. A. Stevens in planning marine machinery. From 1863 to the present time he has been connected with the Bessemer steel and iron manufacture of the United States as follows: In 1863 studied the Bessemer steel manufacture in England; in 1864 to 1865, built and managed the Experimental Works at Troy; in 1865–67 built the 5-ton plant at Troy, and built and managed the Pennsylvania Steel Works and Rail Mill at Harrisburg; in 1868–70 rebuilt and managed the Troy Works (burnt in 1868); in1871–73 built the Bessemer Works at North Chicago and the Bessemer Works and Rail Mill at Joliet; in 1873–75 built the "Edgar Thomson" Bessemer Works and Rail Mill at Pittsburgh, and in 1874–76 built the Vulcan Bessemer Works at St. Louis. From 1871 he has been consulting engineer to the Cambria Iron Company and the Bethlehem Iron Company, and from 1874 to the Lackawanna Iron and Coal Company; is present consulting engineer to ten of the works mentioned, and the construction of their several Bessemer works, and to the Otis Iron and Steel Company and Springfield Iron Company, in the construction of their Open-Hearth Steel Works. He is a member of the United States Government Board to test iron, steel, etc. In 1875–76 he was President of the American Institute of Mining Engineers, and is now a Vice-President of the American Society of Civil Engineers."

Holley was proposed as a member of the Institution of Civil Engineers by the late Sir William Siemens, and was seconded by a long list of well-known and famous men in the profession. They were: F. J. Bramwell, E. A. Cowper, T. R. Crampton, James Ramsden, J. T. Smith, Alexander Brogden, B. Baker, John Hawkshaw, W. H. Barlow, Charles Hutton Gregory, Douglas Galton, I. Lowthian Bell. Some of these men have passed away; they were all his warm personal friends, as were also those who are still living. I should like to read you the following extract from a letter written by one among them whose name was perhaps even better known to a past generation than it is to-day—Sir Charles Hutton Gregory, many years since President of the Institution of Civil Engineers. He writes:

Thirteen years ago I had the pleasure of being one of the proposers of the late Mr. Holley as a member of the Institution of Civil Engineers of England, and having formed a very high opinion of his character and attainments, I feel proud of the fact that we had affiliated this distinguished man to the body of English engineers, and I venture through you to offer to our American brethren my sincere congratulations on their recognition of his merits by a lasting memorial in his honor.

I am, dear Mr. Dredge,
Yours very truly,
CHARLES HUTTON GREGORY,
Past President of the Institution of Civil Engineers.

Holley was also a member of the Iron and Steel Institute of London.

Of the official positions occupied by him, the most important were as follows: In 1875 he was appointed a member of the United States board for testing structural materials; in 1879 he became lecturer on the metallurgy of iron and steel at Columbia College; and in 1878 he was given the degree of Doctor of Laws.

It was early in the summer of 1880 when Holley received the first warning that his career was drawing to a close. Twenty-five years of life at high pressure had told on his constitution, though his energy and magnificent mental powers remained unaffected. Of course, he refused to listen to the intimation that nature, nearly exhausted, gave him. He came over to Europe and completed, under considerable difficulties, his usual arduous round of professional duties on the Continent; then he returned to London in July, tired and suffering from the attack of his old enemy, chills and fever. The malady refused to be driven out by the usual remedies, the symptoms became more complicated, and with alarming rapidity his illness assumed, first a serious and then a most dangerous form, the worst feature of which was that its characteristics baffled the physicians, and made, as events proved, accurate diagnosis impossible.

For weeks he lay between life and death, and you will recall, as if it were but yesterday, the breathless anxiety with which you awaited the receipt of telegrams announcing his condition. By his own insistance, these despatches were made more hopeful than his condition warranted, for his greatest and constant anxiety was to spare his family and friends as far as possible. "Time enough," he used to say, "for them to worry if things go wrong."

It was my privilege to pass many hours by Holley's bedside during this long period of trial, and if my presence, and his knowledge of the constant solicitude of his very numerous friends in London,

afforded him some consolation, his fortitude, his patience, and his calm endurance not only called forth our constant admiration, but taught a lesson I can never forget,—an example few of us can hope to follow. It was very long before the faintest ray of hope threw a gleam upon that sick bed; not until after he had been absolutely condemned by the highest medical opinion. I believe that nothing but his indomitable determination, which never flagged, saved his life. He had from the first a fixed resolution to live, and I think he never once lost confidence. Some of you, at a later time, saw a similar scene enacted, but then, alas! he was able only to keep death at arm's length for a few hours instead of repelling it altogether. It was during this period that I learned the inner aspect of Holley's character. It was no seamy side that was thus exposed to me; but, on the contrary, great and gentle virtues, combined with an absolute trust that all was well, which showed the noble gentleman and the true christian. The closer death approached, the more resigned he grew to the Supreme Will, and yet the more determined to have a longer time secured to him. Not for himself; never a moment for himself; only for those he loved and who were dependent on him. "So far as I am concerned," he would say, "it matters nothing; I have done my best with the powers that I possessed, not for money or ambition, but because I just had to do my best. And it is impossible for me to die for another eighteen months. Those I leave behind me must be provided for, and I can do that easily in a year and a half. Just now I could not afford it."

The light of unselfishness and self-abnegation burnt bright and pure within him, undimned by fears for the future, undisturbed by any weak clinging to the world. If Holley, throughout his life, rejected dogma, it was because his exalted spirit found a surer support in broader views that were more harmonious with his character; and this support never once failed him in the darkest hour. For my part, I can truly say that I hope, when each of us is called away, we may accept the summons as confidently and cheerfully as Holley did, not only during his first illness, but when the end came.

I have insisted upon Holley's marvellous unselfishness during this dreadful period, and I cannot help telling you an incident that illustrates to what an extent he was able to carry this great quality. The *Engineering and Mining Journal* of August 7, 1880, published a telegram that had been despatched from London a few days before. Holley's family and friends were awaiting with eagerness and dread

every message that was sent to them about his condition, and I can imagine the joy with which Dr. Raymond penned the announcement of this despatch. He said: "We reserve to the last the best piece of intelligence, namely, a telegram of the 4th of August, saying, 'Doctors authorize telegraphing improvement more marked.'" I should like to tell you the story of that telegram. For many days hope had been sinking out of sight; consultation after consultation had taken place, and each bulletin was more discouraging than the previous one. The unchanging cheeriness of Holley never failed, but his strength ebbed visibly with every hour. On the morning of the 4th of August a new consultation was held, the most eminent specialist in London, since dead, having been called in. As the doctors stood around the bed of the sick man, speaking with bated breath, a faint smile played over the thin, pale face. You remember Holley's smile. He knew quite well what their opinions were, and seemed less affected than any one else present. Afterwards, in another room, the eminent specialist passed his inexorable sentence: "All that human skill can do has been done; the duration of the patient's life is now a question of a few hours." You may imagine what my feelings were when I returned to Holley's bedside. Presently he whispered to me: "Send them a telegram." "What shall I tell them?" said I. "That the doctors see a marked improvement," he replied. I said: "Can I tell them that, Holley?" "Certainly," he replied; "telegraph it anyway; if you can stand it I can." I sent the telegram Dr. Raymond published, and the remarkable thing was that this was the truest bulletin which had been sent you for a long while. Almost from that hour he began to mend, and improved steadily from day to day. Was not this incident most thoroughly characteristic of the man who, in the direst extremity, thought chiefly of sparing you, and felt that he could "stand" the responsibility of the deception if he was able to give you a few more days of delusive hope?

The time came when there was no need to send any more deceptive telegrams, and it was a great occasion to him and his friends when he was able to leave his room and be carried down stairs. His recuperative powers now stood him in good stead, and he was soon able to look forward to his return home. During the short period that elapsed between his convalescence and his departure, he resumed his interrupted duties, and was able to settle to his satisfaction the work which he had started from New York, six months before, to accomplish.

Thus ended happily what was for so many of us a great crisis. The eighteen months' respite he had fought for so valiantly was accorded to him. How well he used them others know better than I. This may be the place to put on record the fact that during his illness he conceived the idea of his last, and one of his most important, improvements in the Bessemer process,—that of the detachable converter shell. This was patented in 1881, and after his death was purchased by the American Bessemer Association. In England this patent has not found a large application, but, I believe, in the United States it has been extensively introduced, and has helped still further to increase the remarkable production of the various steel works he designed and organized. "If I was to die at that time," said Holley, "I wanted to do something to enable me to die with credit."

It was with gratitude and thankfulness that his friends bade him farewell when he started for his homeward journey in the fall of 1880; not fully restored, but still snatched, as it seemed, from the grave, and given back to those for whose sake he had battled through his long illness,—his family and friends. The winter, spring, and the first months of the summer of 1881 he passed with you; therefore, you know better than I do that the lamp never burned brighter, though to those who watched him closely the flame was somewhat fitful, for the oil was nearly spent. Perhaps you may be interested in the following extracts from letters I received from him in 1880. They contain nothing of importance, but to me they have a special value, and they indicate how far he was restored to apparent health and to real energy. The first was written from Liverpool on October 8, 1880, just before he sailed for New York, and about a month after he had left the doctor's hands. During this time he had prepared a paper on "The Adaptation of Bessemer Plant to the Basic Process," which was read before your Society of Mechanical Engineers at their annual meeting, and he had visited the various works in England, where business awaited him. He wrote:

"Enclosed find proof of my paper to be read November 24th. . . . . I am very glad I made the run to Glasgow and Eston, as I looked up much of interest and received no harm as far as I know of. I hope that you will see much of my best work in *Engineering* during the next six months."

The second letter, dated October 20th, was written from his home in Brooklyn:

"I have been home an hour, and I hasten to tell you, by to-morrow's early steamer, that I found my family well, that I gained strength on the voyage, and I am feeling very well. . . . . I found my wife and children on the wharf. They were surprised to see me looking so unusually well, and glad enough to see me so, or to see me at all in fact, and they joined me in expressions of gratitude for my deliverance from the King of Terrors. . . . . As I settle down in my place at home, it sometimes seems as if I had not been away (though it will be six months to-morrow since I sailed), and my illness had been a bad dream."

The numerous letters I received from him between this latter date and the following July, when he came to England, all breathe the same spirit of thankfulness for his recovery, the same interest in the work he had to do, his hopefulness for a prolonged and useful future. It is evident, however, he must have been conscious that his physical elasticity had failed, for while he carried on his work as thoroughly as ever, there were not wanting proofs that he knew the necessity of caution, and he made plans for securing leisure and rest which a short time before he would have deemed entirely unreasonable. I think that he missed few, if any, of your professional meetings, or the banquets, which are one of their pleasant features; and on those occasions his eloquence was never loftier, nor his humor more keen and delicate, as witness his remarkable speech at the Hartford meeting of your Society of Mechanical Engineers in 1881.

With the summer of that year came the perfected plans for his annual professional European visit, the last, he had assured me, twelve months before, that he intended to make—a fateful prophecy. But on that occasion he departed from his usual custom, and relaxation formed a part of his scheme, to insure which his whole family travelled with him. In this connection I do not hesitate to read to you extracts from a letter he wrote me on the steamer "Germanic" before his arrival at Liverpool. The letter is dated July 17, 1881:

"We have had a very smooth passage, and everything delightful. . . . . We intend to go to Scotland for two weeks, and then spend two or three days in London on the way to Switzerland, which we wish to reach as soon as possible so as to do it before cold weather. Charles Macdonald, one of our prominent engineers and bridge-builders, and no end of a good fellow, is with us, and I want him to meet some of the boys. I shall write you what day we shall arrive in London. And I am going to put you to the trouble of ordering for me an A 1 dinner, with smoking-room, and inviting yourself and a few more, ... We must have that dinner."

One of the main objects Holley had at heart in planning his European trip of 1881, was the realization of a scheme, the first intima-

tion of which was conveyed to me in the letter I have just quoted. The initiation of this scheme was, I believe, wholly due to himself; and certainly no one could so well understand its great importance. Speaking after an interval of nine years, I may be in error in some details, but I think that, between his return to America, in 1880, and his last visit to Europe the following summer, the idea had occurred to him to bring about closer relations between the members of the engineering profession in England and in the United States. No one knew better than he did the value of personal intercourse, or the rapidity with which the mists of indifference, prejudice, or ignorance are dispelled under the genial influence of personal acquaintance. Perhaps, too, Holley, who was so justly proud of all he had accomplished, was anxious that Englishmen should see for themselves what they only knew from his lips, and the truth of which they half doubted; in any case, I know that he chafed under the recollection that his statements had more than once been received with a scarce-concealed spirit of incredulity. These personal motives may have somewhat influenced him; but his chief reason in wishing to bring English and American engineers in closer touch with one another, was the lofty one of giving to both nations more light and sympathy. I understand that his proposal was carefully discussed by a joint committee of your societies, and that a magnificent programme of reception was prepared. Then Holley and his friend, Mr. Charles Macdonald, whose eminence as a civil engineer is almost as fully recognized in Europe as it is in America, were asked to act as the delegates of your societies, to convene in London such a meeting as they thought most representative and useful; and to lay before this meeting your proposals, that they might ascertain how such an invitation would be received.

The first meeting, Holley very naturally decided, should take the form of a dinner, and it was this dinner which is referred to in the letter I just now read. Of course, I knew nothing of the details at that time, but I judged that something of importance was on foot, for until Holley arrived in London, he showered letters upon me always on the same topic. When we met, he was very full of this scheme, and it was hard not to encourage him, or even not to share his enthusiasm. I felt that a great disappointment awaited him, and that the project he so much cherished was doomed to failure. The time had not come for such a union; but Holley, in the noble generosity of heart which had conceived and elaborated the proposal, could not understand that it might be received with coldness.

The dinner took place, and, with the exception of Holley, Mr. Macdonald, and myself, no one understood why he had been invited. There were but few guests, and to them, after the close of dinner, Holley unfolded his scheme; told, in his eloquent and winning way, how warmly you desired to welcome your brother engineers from London; how no limits would be set to your hospitality; how your factories would be thrown open, your professional experience and information freely given. But it became at once evident that the scheme was impracticable, and would have to be abandoned.

I am not seeking to blame those gentlemen who thus met Holley's warmth with seeming coldness, and chilled his generous enthusiasm. The time had not come. But Holley, in his disappointment, did not, perhaps, make full allowance for the want of appreciation displayed, and was very sick and weary in his failure, which greatly saddened the few remaining months he was to pass in England. And I know how loudly the echo of his failure resounded through the United States. Thanks to yourselves, the memory of that time was blotted out last year, when you responded so fully to the brotherly invitation sent you by the Institution of Civil Engineers, who, you were pleased to say, extended to you true hospitality and friendship with both hands. I felt the deepest interest in the result of that invitation, and I strove earnestly, in my small way, to help to make your visit a success, for I felt that it was a sacred duty which I owed to the memory of our dear friend to assist in bringing to pass what he had so earnestly longed for. Your visit to England last year was the first fruit of Holley's labor of 1881. Are we not all filled with the same vain longing that he might be among us now to see the further result of his work; to act as host to the foreign guests now assembled on your shores; to hold us once again by the spell of his eloquence?

The plans which Holley had formed before he left New York of obtaining rest and change by a European tour, were never carried into execution; probably his habits of continuous work were too strong to be broken up. At all events, while his family set out to complete the programme, Holley remained in England for awhile, and then made his accustomed round of the steel works in France, Germany and Belgium. He was back again in England in time to attend and take a part in the meeting of the Iron and Steel Institute. On October 20, 1881, he wrote to me:

"Very sorry to have missed your call. I have not felt quite up to going north

yet, and probably shall not go before the middle of next week. I naturally overdid the thing a little at the meeting, and have had some mild returns of chills and fever, but Dr. Hare thinks he will break it up in a few days. I shall not go to Italy at all, but shall remain in England until near the end of November and then meet my family in Paris. I mean to get out that Bethlehem supplement for you before many days, but what with quinine and rudimentary chills I have felt fit for little."

Now Holley's work was done; his plans for journeying north were only partially carried out, and he returned to London early in November to meet his family. With the mist and gloom of the last days of autumn began the closing scene of this bright career, saddened by a great and unexpected anxiety and grief. During the visit of his family to Rome, one of his daughters contracted a fever that developed with alarming symptoms almost as soon as they were reunited in London, and for several weeks she lay a victim to a severe attack of typhoid. A fortunate chance had made me recommend to Holley a hotel owned by a gentleman whose name you have all heard; a man I am proud to call my friend; who filled with distinguished ability the high position of Lord Mayor of London, and who was so largely instrumental in making the Paris Exhibition of 1889 an international success. I refer to Sir Polydore de Keyser. During many weary weeks, when Holley's final illness, which had now taken its final hold upon his system, was intensified by anxiety at the alarming condition of one of the three persons to whom his life had been devoted, he found in Sir Polydore and Lady de Keyser two of the truest and most devoted friends. It was greatly due to their unceasing solicitude that Holley's daughter was restored to convalescence before he was ordered hurriedly to quit England. To obey this order meant separation from his family, for his daughter was unfit to travel. But Holley realized that this was his only chance, and slender as he knew it to be, he did not hesitate for a moment. But he would have refused utterly if Providence had not thrown in his way those two good friends, who, after this lapse of years, still cherish his memory with lively affection, and in whose hands he left his family with full confidence.

Holley left London for the last time a few days before the Christmas of 1881. It was one of those dark and gloomy mornings which we know only too well, when the city is swallowed up in chill mists, and mid-day is only a sickly twilight. When we shook hands as the train started, he said to me: "This is, indeed, good-bye." And then I knew that we should never meet again. I have always regarded this last journey of Holley's, taken suddenly at so bad a

season of the year, and in the state of health he knew himself to be, as a final act of unselfishness which seemed natural to him, but of which few people are capable. He felt that he would rather separate himself from those he loved, than add to their anxieties, and at any cost that he must grasp at the small chance which the better climate of America might give him; but all the same he knew that the end was come.

The sad heroic details of the closing scene you all know well, but I will venture to reproduce the story here as told in Dr. Raymond's eloquent language:

"What a Sabbath followed! The quiet chamber where he lay, brave and cheerful, watching the strength that was ebbing away, as he might have watched and measured the current of a motive power; the storm-beaten ship that steamed up the harbor, bearing those he loved best; the race with death; the arrival of wife and daughters, just too late. What insupportable wringing of all hearts, except his, who recognized with calm impartiality when the machinery of life was about to stop, and whose only anxiety was for those who were sailing unawares towards the shadow of death! 'Make it as easy for them as you can,' he had earnestly said to a faithful friend." *

The widespread sorrow that descended on us with Holley's death like a dark veil, has been lifted by the gentle hand of time, and looking back we are able to see, in true perspective, the great achievements of his busy life; the work that was left unfinished when death called him. It must have occurred to many of us that, after all, it was better so; better for him to have departed from us while still in the vigor of intellect than to have lingered until his powers failed and the light of his spirit burnt low. He completed in his lifetime more than enough to fill the record of many ordinary lives, and he remains enshrined in our hearts, brilliant and noble. Can we desire more? Can any one of us hope for so much?

I prefer to remember Holley rather as a man of talent than of genius; that is, if we accept the strict meaning of the word "genius," and regard it as a special gift of arriving always at a desired aim, by intuition—in two words, "inspired instinct." Talent, on the other hand, depends on high mental training and a perfect command of all the faculties; on a great strength of intellect, and on a peculiar aptitude for being moulded and directed to specific ends and to valuable purposes. Does not this apply in all respects to Holley's mind? His great natural gifts were steadily and systematically trained by

* *Eng. and Min. Jour.*, vol. xxiii., p. 62, Feb. 4, 1882.

self-imposed discipline and by the sterner force of circumstances. Yet, if we accept the great philosopher's dictum, that genius consists of a power to take infinite trouble, then we must ascribe the gift of genius to Holley in a high degree; for in all things, large and small, no amount of trouble was too great for him, provided that what he did was well done. But genius is nearly always ill-balanced, and the great qualities of one so endowed are generally accompanied by petty weaknesses that only too often make these special gifts a burden to the possessor and to those about him. Holley was great all round; as engineer, as inventor, as journalist, and as orator. Equally great was he in his private life. Admired by all those who were his acquaintances; beloved and equally admired by those who had the privilege to be his friends; a devoted husband and a devoted father; one whose only thoughts in moments of extremity were for others but never for himself. No; let us ascribe to Holley the highest, the most brilliant talents; but do not place it on record that he was a man of genius, for in so doing we imply the existence of weaknesses that had no place in his great nature.

Your faithful and enduring friendship has found graceful expression in marble and in bronze; in a statue which shall remain to speak to future generations of their great engineer who revolutionized the most important industries of America. But this memorial which you have erected is only the outward and visible sign of the spiritual memorial set within your hearts, to be kept there sacred through life—and beyond it. Many of you have seen carved over one of the portals of St. Paul's Cathedral the epitaph of its architect: *Si monumentum requiris circumspice.* So you must look around you for the monument of Holley. It stretches in a vast and ever-growing net-work over your country; from east to west, from north to south, wherever railways are, his monument lies beneath you, as you travel over the lines of steel. You behold it in the vast piles of buildings devoted to the production and the manufacture of steel. The memorial to his great energy and intelligence is made visible in pillars of cloud by day and in pillars of fire by night in your greatest metallurgical centers. The development of your most important mineral resources; the contentment and prosperity that wait upon the vast army of laborers in the field where Holley first broke ground; the wealth that has attended those who were far-seeing and bold enough to support him; to no small extent the marvellous growth of your great country, that causes us, the slow-moving nations of the Old World, to stand and gaze in wonder. All these great things

together compose Holley's monument; so if you seek it, your gaze must not be restricted to narrow limits, such as those that bound the matchless pile reared by our great architect more than two centuries ago; it must range from the Atlantic to the Pacific; from the Gulf of Mexico to the Arctic circle. On every piece of Bessemer steel that is converted to the service of the arts and industries in the United States might be engraved the words he used to write upon the many drawings he made in London during his first youthful visit, with such a rapid and unerring hand: "Holley Fecit."

I have completed the work your generosity has imposed on me, and, with all imperfections, this address must stand in the archives of your societies, to bear testimony to my unfitness for the task. In one respect I feel a full measure of satisfaction; I have brought you from beyond the sea the proofs of how Holley was esteemed by strangers—a gift of precious pearls of human sympathy, worthy to be laid upon the base of your memorial column. Would that they were linked together by a golden cord instead of being strung upon a thread of baser metal sublimated from very common clay.

Mr. Dredge's eloquent, sympathetic, and discriminating address was received with applause, and with an absorbed and appreciative attention more significant than applause. At its conclusion the assembly proceeded to Washington Square, where the memorial, still hidden from view by the drapery of an American flag, was to be disclosed and formally turned over to the custody of the authorities of the city of New York, represented on this occasion by President Gallup, of the Board of Park Commissioners.

After music by the band of the Seventh Regiment, of the New York National Guard, Mr. James C. Bayles, a representative upon the Holley Memorial Joint Committee of the American Society of Mechanical Engineers, and ex-President of the American Institute of Mining Engineers, addressed Mr. Gallup substantially as follows:

"In the name of the Holley Memorial Committee, I surrender to you, and through you to the city of New York, which you here represent, this symbol and expression of civic fame and human affection. We who have placed it here desire it to express not only our love and esteem for a great engineer and a dear friend, but also our profound conviction that the achievements of genius and devo-

tion in the field of science and industry are not less worthy of monumental commemoration than the successful labors of the statesman or the victories of the soldier. In this spirit we trust that the Holley Memorial will be accepted and preserved by the city of New York for the instruction and encouragement of generations."

At the close of Mr. Bayles's remarks, the Memorial was unveiled by the hand of Master Alexander Holley Olmsted, a boy of six years, the grandson of Mr. Holley, and was received on behalf of the city of New York by President Gallup, who spoke as follows:

"It is indeed fitting that in this country, where genius and invention are triumphant, our citizens should turn aside now and then from their labors to pay just tribute to those who have made her great. Among them truly was he who has been so honored to-day as one of the greatest of engineers, and of whom it can well be said, borrowing the words from another, that though he ever strove to hide his light under a bushel, yet it was among the strongest and brightest lights that ever burned. I would that all our parks and squares might contain such object-lessons as this, not alone because of the art that fashioned it, but because of the energy, enthusiasm, intensity of purpose, great honor and just achievements of him whom it calls to mind. Your chairman has spoken of the beneficent victories of peace. In the same spirit I accept this bronze on behalf of the city of New York. Once before we struck hands with a people to whom we owe the beacon light that lighted the way of our guests to this port; and so I may extend to them a like welcome, that this memorial may serve as another emblem of the good-will of nations."

PROCEEDINGS OF THE FIFTY-NINTH MEETING, CLEVELAND, O., JUNE, 1891.

Local Committee.

Fayette Brown, *Chairman;* J. K. Bole, *Vice-Chairman;* E. W. Ogleboy, Secretary.

Sub-Committees.

Ladies Committee.—Mrs. Pechin, Mrs. Hanna, Miss Hilliard.

Reception.—Amos Townsend, W. R. Warner, M. A. Hanna, G. H. Ely, A. A. Pope, S. A. Fuller, R. R. Rhodes, D. B. Chambers, E. C. Pechin, E. W. Ogleboy.

Excursions.—S. H. Chisholm, L. A. Roby, R. F. Jopling, George Bartol, Alexander E. Brown.

Banquet.—L. C. Hanna, William Edwards, J. B. Zerbe.

Finance.—John Tod, Samuel Mather, S. P. Ely, C. W. Bingham.

Hotel Headquarters.—The Stillman.

The opening session was held on Tuesday evening, June 2d, in the hall of the Young Men's Christian Association. The meeting was called to order, in behalf of the Local Committee, by Mr. G. H. Ely, who introduced Hon. W. G. Rose, the mayor of the city. Mayor Rose delivered a brief but cordial address of welcome, to which an appropriate response was made by President Birkinbine, who referred to the former meeting of the Institute at Cleveland, in 1875, and the progress, both of this society and of the city of Cleveland, since that pleasant and well-remembered occasion.

An address on Modern Engineering in Egypt, illustrated with lantern-views, was then delivered by R. W. Raymond, of New York City.*

The second session was held at The Stillman, on Wednesday morning, June 3d, when the following papers were read and discussed:

Electricity in Welding and Metal-Working, by A. B. Wood, Detroit, Mich.

A Review of the Iron-Mining Industry of New Jersey, by J. C. Smock, Trenton, N. J.

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* This address, consisting, in the main, of comments upon the views, has not been reduced to writing.

The First Iron Blast-Furnaces in America, by W. H. Adams, New York City.

The remainder of the session was occupied with a discussion of blast-furnace practice, suggested by the paper read by Mr. Gayley on that subject at the New York meeting of the Iron and Steel Institute, October, 1890.

At the third session, held at The Stillman on Wednesday evening, the discussion of blast-furnace practice was continued, after which Mr. Alexander E. Brown, of Cleveland, made an address on the handling, by machinery, of ore and fuel in large quantities.*

At the fourth session, the Secretary presented, in continuation of the discussion on blast-furnace practice, the following paper:

Some Experiments on Blast-Furnace Gases, by Jasper Whiting, South Chicago, Ill.

A discussion of the paper of Mr. T. D. Ledyard (New York meeting, Vol. XIX.) on Ontario Magnetites, followed.

Also, a discussion of the paper of Mr. H. H. Campbell (New York meeting) on the Physical and Chemical Equations of the Open-Hearth Furnace.

Also, a statement from Prof. J. W. Langley in continuation of his paper (Pittsburgh International Session, Vol. XIX.) on International Standards of Iron and Steel Analyses, after which the following papers were read:

Aluminum in Steel Ingots, by Prof. J. W. Langley, Pittsburgh, Pa.

Manganese in Cast-Iron, by W. J. Keep, Detroit, Mich.

The following papers were then presented in printed form, or read by title:

Construction of Details for a Modern Lixiviation-Plant, by Charles A. Stetefeldt, San Francisco, Cal.

Geological Notes on the Manganese Ore-Deposit of Crimora, Va., by Charles E. Hall, Chicago, Ill.

A Compound-Plunger Hydraulic Pump, by Earnest R. Woakes, Tolima Mines, Colombia.

Notes on Some of the Magnetites of Southwestern Virginia and the Contiguous Territory of North Carolina, by H. B. C. Nitze, Baltimore, Md.

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* Not furnished for publication.

Methods of Working and Surveying the Mines of the Longdale Iron Company, Virginia, by Guy R. Johnson, Longdale, Va.

Stone-Coal in the Lead Blast-Furnace, by James W. Neill, Leadville, Colo. (with a communication, in discussion thereof, from A. S. Dwight, Pueblo, Colo.).

Explosions from Unknown Causes (discussion of the New York paper of Mr. J. C. Bayles, Vol. XIX.), by George R. Greene, East Orange, N. J.

Chinese Silver-Mining in Mongolia, by H. F. Dawes, Englewood, N. J.

The Alluvial Tin-Deposits of Siak, Sumatra, by Charles M. Rolker, New York City.

The Precipitation of Metals from Hyposulphite Solutions, by Charles A. Stetefeldt, San Francisco, Cal.

The Mount Morgan Mine, Queensland, by T. A. Rickard, Melbourne, Australia.

The Refining of Sulphides Obtained in the Lixiviation-Process with Hyposulphite Solutions, by Charles A. Stetefeldt, San Francisco, Cal.

Experiments with the Imperatori Process at Croton Magnetic Mines, New York, by J. B. Nau, New York City.

Sampling Ores Without Use of Machinery, by William Glenn, Baltimore, Md.

Sinking Through Wet Gravel and Quicksand near Norway, Michigan, by William Kelly, Vulcan, Mich.

The Direct Determination of Aluminum in Iron and Steel, by Thomas M. Drown and Alexander G. McKenna, Boston, Mass.

After the passage of suitable resolutions of thanks for courtesies received, the meeting adjourned.

Members and Associates Elected.

The following persons, duly recommended by the Council, were elected as members and associates at the several sessions of this meeting:

Members.

Walter A. Ballard,	Pittsburgh, Pa.
Ygnacio Bonillas,	Magdalena, Mexico.
Frank Buhl,	Sharon, Pa.
John B. Coryell,	Corning, N. Y.
Edward Doud,	Sheffield, Ala.
J. W. Dougherty,	Steelton, Pa.
C. H. Ferry,	Chicago, Ill.
George H. Ely,	Cleveland, O.

Willard Fuller,	Cleveland, O.
William M. Given,	Buena Vista, Va.
William Kennedy,	Austin, Tex.
William J. Kerr,	Roanoke, Va.
Clement Le Boutillier,	Pittsburgh, Pa.
R. G. E. Leckie,	Middleton, N. S.
D. McVichie,	Iron Belt, Wis.
E. Kennard Mitting,	Chicago, Ill.
Frank Nason,	New Brunswick, N. J.
William G. Parke,	Scranton, Pa.
William B. Parsons,	New York City.
Gerard J. Partington,	Whiteburn Mines, N. S.
Frederick D. Power,	Melbourne, Australia.
Joseph Thomas Pullon,	Pachuca, Mexico.
Axel Sahlin,	New York City.
G. A. Sonnemann,	Wardner, Idaho.
Erastus Wiman,	New York City.

ASSOCIATES.

William S. Pilling,	Philadelphia, Pa.
E. H. Sanborn,	Rosemont, Pa.

M. R. Hunt, Ashland, Wis., having been elected formerly an associate by reason of a clerical error in the proposal of his name, as submitted to the Council, was unanimously made a member, in correction of the said mistake.

EXCURSIONS AND ENTERTAINMENTS.

The ladies of the party were most assiduously and agreeably entertained with drives, social receptions, etc., organized by the Cleveland Ladies' Committee.

Thursday evening, June 4th, was most delightfully occupied by the dinner, given at The Stillman.

The excursions connected with the meeting consisted mainly of visits to the many works, ore-docks, water-works, and other points of professional interest in and about the city (a projected trip on Lake Erie, in one of the new ore-carriers, being necessarily abandoned by reason of a labor-strike, which made it impossible to prepare the vessel intended for this purpose). These visits were made by large or small parties or by individual members, in accordance with the liberal invitations extended through the Local Committee by the various Cleveland companies, societies, firms, and individual proprietors. The following list of formal invitations received does not exhaust the catalogue of the hospitalities actually bestowed:

Civil Engineers' Club; Union Club; St. John Typobar Company; Lake Shore Pipe Foundry; Otis Steel Company, Limited;

H. P. Nail Company; Cleveland City Forge and Iron Company; Union Steel Screw Works Company; Brown Hoisting and Conveying Company; American Wire Company; Baackes Wire Nail Company; King Iron Bridge and Manufacturing Company; Brush Electric Company; Cleveland Rubber Company; Cleveland Rolling Mill Company; Broadway and Newburgh Street Railroad Company; East Cleveland Street Railroad Company; Cleveland City Cable Railway Company; Cleveland Shipbuilding Company; Globe Iron Works Company; Walker Manufacturing Company; Avery Stamping Works.

At all these places the badge furnished by the Local Committee to visiting members was a sufficient introduction.

Registry.

The following names were registered in the office of the Local Committee at hotel headquarters. As is usually the case when an Institute meeting is held in a large city, the registry is doubtless smaller than the actual attendance, many members arriving late, attending sessions only, lodging at other hotels, stopping with friends residing in the town or the suburbs, etc., and neglecting to register their names.

Guilliaem Aertsen.
W. A. Ballard.
George Bartol.
John Birkinbine.
W. H. Blauvelt.
Lee Burt.
J. H. Cremer.
George A. Dean.
John S Fackenthal.
E. M. Ferguson.
S. T. Ferguson.
Edward L. Ford.
G. C. Gardner.
William Gerhauser.
Charles B. Green.
G. C. Hewett.
H. D. Hibbard.
Theodore M. Hopke.
F. F. Hunt.
J. M. Jaycox.
J. E. Johnson.
Eliot A. Kebler.
Hugh Kennedy.
Julian Kennedy.
W. J. Keep.
Charles Kirchhoff.
John W. Langley.
T. D. Ledyard.
Thomas H. Leggett.
Aug. W. Newell.
E. C. Pechin.
W. H. Pettee.
Frank A. Ray.
R. W. Raymond.
William H. Rea.
S. B. Robbins.
Walter Russell.
Edward H. Sanborn.
J. C. Smock.
H. C. Spaulding.
C. E. Stafford.
John Stevenson, Jr.
Ambrose Swasey.
John M. Thomas.
S. Whinery.
W. H. Wiley.
A. B. Wood.
James E. York.

PROCEEDINGS OF THE SIXTIETH MEETING, GLEN SUMMIT, PA., OCTOBER, 1891.

LOCAL COMMITTEE.

Eckley B. Coxe, Drifton, *Chairman;* Irving A. Stearns, *Secretary;* and Messrs. W. A. Lathrop, G. M. Williams, T. H. Phillips, W. J. Richards, H. S. Reets, C. M. Conyngham, L. D. Shoemaker, H. B. Hillman, Paul A. Oliver, A. Mitchell, C. P. Hunt, Elmer H. Lawall, Stanley Woodward, W. M. Shoemaker, A. H. Dickson, J. Roberts, Jr., A. J. Davis, J. H. Bowden, R. Van A. Norris, R. B. Ricketts and W. H. Sturdevant of Wilkes-Barre; W. T. Smith, A. H. Storrs, Clarence D. Simpson, James Archbald, Samuel Hines, E. S. Moffat, F. E. Platt, and Rufus J. Foster, of Scranton; Thomas McNair and T. D. Jones, of Hazleton: W. G. Payne and Daniel Edwards, of Kingston; Gen. Paul A. Oliver, Oliver's Mills; Jno. B. Law, Pittston; J. C. Haddock, Glen Summit; Jno. Markle, Jeddo; A. P. Blakslee, Delano; Albert Lewis, Bear Creek; Geo. T. Morgan, Nanticoke; and M. H. Kemmerer, Mauch Chunk.

Headquarters for the meeting and all sessions were at the Glen Summit Hotel.

The opening session was held on Tuesday evening, October 6th. Hon. Eckley B. Coxe, Chairman of the Local Committee, called the meeting to order, with a few words of cordial welcome, to which President Birkinbine made a suitable response, recalling the circumstance that the name of Mr. Coxe was the first of the three names* attached to the circular which led to the organization at Wilkes-Barre in May, 1871, of the American Institute of Mining Engineers. He saw among those present at this meeting at least eight of the original thirteen, whose number had now grown to more than two thousand. As an evidence of the wide influence of the Institute, he called attention to the list of candidates who would be recommended for election to membership at this meeting, nearly one-third of whom were residing in foreign countries and could not expect to share the social pleasures of the Institute meetings, yet appreciated with enthusiasm the value of its *Transactions*. In concluding his introductory remarks, Mr. Birkinbine alluded to the circumstance that, apart from excursions connected with meetings at Philadelphia, this was the eighth visit of the Institute to the region of its origin, meetings

* Eckley B. Coxe, R. P. Rothwell and Martin Coryell.

having been held twice at Wilkes-Barre, twice at Bethlehem and once each at Easton, Hazleton and Scranton. Nevertheless, there would be on this, as on each preceding occasion, much to see that would prove novel, as well as instructive. Mr. Birkinbine then proceeded to deliver the Presidential Address, on the Fuel-Supply of the United States.

After the election of a large number of members and associates (see list below) the following papers were read and discussed:

A Hand-Telescope for Stadia-Work, by Robert H. Richards, Boston, Mass.

Results of Stream-Measurements of the U. S. Geological Survey, by F. H. Newell, Washington, D. C.

The second session was held Wednesday morning, October 7th, when the following paper was read and discussed:

The Florence Oil-Field, Colorado, by George H. Eldridge, Washington, D. C.

The President read the following telegram from F. J. V. Skiff, Chief of the Department of Mines and Mining in the Chicago Columbian Exposition:

"I am directed by the Director-General of the World's Columbian Exposition to congratulate your distinguished organization on its sixtieth convention, and invite the co-operation of the American Mining Engineers, to the end that the science and industry to which you are devoted may take its proper rank in demonstrating the world's progress."

A discussion then took place on the topic of the magnetic concentration of iron-ore, in connection with which the following papers were presented:

The Use of Magnetic Concentrates in the Port Henry Blast-Furnaces, by N. M. Langdon, Port Henry, N. Y.

Practical Results in the Magnetic Concentration of Iron-Ore, by W. H. Hoffman, Brewster, N. Y.

Note on Sampling Iron-Ore, by E. K. Landis, Pottstown, Pa.

The Determination of Iron in the Tails from Magnetic Concentration, by E. K. Landis, Potttstown, Pa.

During the discussion, which occupied the remainder of the session, the Secretary presented also a communication on the subject from Clemens Jones, Hokendauqua, Pa.

The third session was held Wednesday afternoon.

B. E. Fernow, Washington, D. C., made some remarks on the

proposed government tests of timber, as calculated to furnish information valuable to engineers.

The following papers were read and discussed:

Tests and Requirements of Structural Wrought-Iron and Steel, by A. E. Hunt, Pittsburgh, Pa. (Discussed by Wm. H. Morris, Wm. Kent, Oliver Williams, C. B. Dudley, Eckley B. Coxe and others.)

Centrifugal Ventilators, by R. Van A. Norris, Wilkes-Barre, Pa. (Discussed by A. H. Storrs, J. C. Platt, T. D. Jones and others.)

Before the adjournment of the session, Messrs. A. E. Hunt, J. F. Holloway and J. H. Harden were appointed a committee to prepare suitable resolutions in acknowledgment of the services and hospitalities of the Local Committee and in recognition of the peculiar significance of this meeting of the Institute, nearly twenty-one years after its organization, in the vicinity of the locality of that event. (The report of this committee, subsequently adopted by acclamation at the "camp-fire" of October 8th, is given below.)

The fourth and concluding session was held on Wednesday evening, when the following papers were read and discussed:

Notes on a Novel Cable-Transfer for Railroad-Cars, and the Use of the Patent Locked-Rope, by E. G. Spilsbury, Trenton, N. J.

Electric Locomotives in German Mines, by Karl Eilers, Berlin, Germany. (Discussed by H. C. Spaulding and others.)

The Utilization of Puddle- and Re-heating-Slags for Paint-Stock, by Axel Sahlin, New York City.

The remainder of the session was occupied with the discussion of the Preparation and Utilization of the Small Sizes of Anthracite Coal, opened with a paper on that subject by Eckley B. Coxe, Drifton, Pa.

In connection with this discussion, a paper was presented on The Utilization of Anthracite Waste by Gasification in Producers, by W. H. Blauvelt, Philadelphia, Pa.; also, a brief communication from F. M. F. Cazin, New York City, concerning the manufacture of coke from mixed anthracite and bituminous "smalls;" and a paper was announced by R. J. Foster, Scranton, on the Use of the McClave Grate and Argand Steam-Blower. The discussion was continued by Messrs. E. G. Spilsbury, Wm. Kent, R. J. Foster, J. C. Haddock, E. S. Moffat, E. V. D'Invilliers, W. H. Morris, J. F. Holloway, C. B. Dudley and others, and at its close the meeting was adjourned.

In addition to the papers above mentioned as read, the following were presented in printed form, but not discussed:

Electricity in Mining, as Applied by the Aspen Mining and Smelting Company, by M. B. Holt, Aspen, Colo.

The Manufacture of Liquid Sulphurous Acid in Upper Silesia, by Karl Eilers, Berlin, Germany.

A Chinese System of Gold-Milling, by Henry Louis, Singapore, Straits Settlements.

Cord-Wood in the Matting Blast-Furnace, by Herbert Lang, Mineral, Idaho.

Apparatus for the Manipulation of Iron and Steel Plates During the Process of Finishing, by Gram Curtis, Pittsburgh, Pa.

The Handling of Ingots and Moulds in Bessemer Steel Works, by Gram Curtis, Pittsburgh, Pa.

Tandem-Tanks for Hoisting Water from Flooded Slopes, by J. H. Bowden, Wilkes-Barre, Pa.

And the following papers were read by title:

The Bendigo Gold-Field, by T. A. Rickard, Allemont, Isère, France.

Mining in Honduras, by W. A. Thacher, Leadville, Colo.

Photographing and Co-ordinate Surveying, by Henry M. Stanley, Nogales, Arizona.

A New System of Ore-Sampling, by H. L. Bridgman, Blue Island, Ill.

Notes on the Iron-Ores of Danville, Pa., with a Description of the Long-Wall Method of Mining Them, by H. H. Stok, South Bethlehem, Pa.

Report of the Special Committee.

The following report of the Special Committee, appointed at the third session of the meeting, was presented and adopted by acclamation at the "Camp-Fire" on Thursday evening:

Mr. President and Members of the Institute:

In assigning to their committee its duties, you were pleased to remind us of the fact that twenty years ago the Institute was founded at and held their first meeting in Wilkes-Barre, which, as you know, lies at the foot of the mountain and in the beautiful valley of Wyoming, and but a short distance from here. Whether the locating of the twentieth anniversary at Glen Summit, high above the valley in which the Institute had its origin, was by accident or design, we do not know, but it would seem to be significant in more ways than one. The time at disposal will prevent an extended recital of the history of the Institute in these interesting years, but of its growth and progress it may be said, beginning as it did in a modest way, it has gradually increased in membership and in the quality of its publications until to-day it stands figuratively, as it does in reality, on the ground near to where

it started, but on a summit of importance and excellence that looks with commendable pride on its lovely birth-place.

The original call for a meeting was signed by three persons, and the name at the head of the list is the genial chairman of the present Local Committee, Eckley B. Coxe, followed by R. P. Rothwell and Martin Coryell. The last-named gentleman was for two years the secretary of the Society, and the secretary *pro tem.*, curiously enough, was the gentleman who now is and has been for the past seven years secretary of the Institute; and it no doubt is due to the fact of his early training in this line that he has proven to be such an efficient officer. The Institute had the good fortune to have for its first president one of the great pioneer iron-makers of this country, David Thomas. Since then fifteen presidents have filled the office, two of whom served two terms each, and one, our present secretary, served for three.

To speak of the men who, since the first meeting in 1871 to the present, have joined its ranks and contributed to its literature, would be a task your committee could not undertake; but they feel that they would not have fulfilled the obligation imposed upon them in this retrospective view did they not recall to your memory such names as Thomas, Ashburner, Coryell, Holley, Gowan, Henrich, Firmstone, Cheever, Captain Jones, James Park, Jr., General Pleasants, Henry Worthington, and others, the records of whose achievements still linger in your memory, and who have gone to join the "great majority."

A recital of the history of the Institute would be incomplete that did not recognize the lasting obligations we are all under to that faithful, efficient, and industrious secretary of eleven years' standing, who, in the hour of peril, neglecting his personal interests, saved from the fire the treasures you intrusted to his care, Thomas M. Drown. If you refer to the call for the first meeting, issued in April, 1871, in which so well was set forth its objects and aim, and will then turn to the nineteen volumes of its *Transactions* which adorn your library, we are certain you will concede that not only were the foundations well prepared by the early pioneers, but as well there has been built a noble structure of engineering knowledge and experience which, in the coming years, will be helpful to all who shall come after us. It is one of the pleasantest features of this gathering that we have with us at least six members who were a part of that first meeting at Wilkes-Barre, Messrs. Coxe, Raymond, Harden, Allison, Oliver, and Stearns, and possibly others.

Resolved, That the thanks of the American Institute of Mining Engineers be tendered to the Lehigh Valley Railroad Company for the generous repetition of favors so often bestowed in the way of transportation over their lines.

Resolved, That the hearty thanks of every man, woman, and child who had the good fortune to participate in the excursion in the Wyoming Valley and the luncheon at Harvey's Lake, is due and thankfully tendered to Albert Lewis, whose thoughtful attentions and bounteous hospitality made the occasion one long to be remembered.

Resolved, That the attention paid the Institute by the press of Wilkes-Barre in the full and accurate report made of its proceedings, merits our warmest thanks.

Resolved, That the Local Committee, who, through their officers, Messrs. Coxe, Stearns and Lathrop, left nothing undone that could have added to the snap of the meeting or the enjoyment of the members and their guests, be tendered our kindest regards and sincerest thanks.

Resolved, That we know we shall want to thank Eckley B. Coxe and his associates for the kindly services that we are sure of receiving at his hands to-morrow, and therefore wish to say thank you now, while we are assembled and have the opportunity for this anticipated pleasure.

Resolved, That thanks be given to General Paul A. Oliver, who so handsomely entertained us at his country-seat, where the cheery brightness and the glowing warmth of his blazing camp-fires were emblematic of the man who, in so many ways, has made the world better and brighter for his being in it, and if he to-night enjoys having us here as we enjoy being here, he is a happy man.

Members and Associates Elected.

The following persons, duly recommended by the Council, were elected at the several sessions:

MEMBERS.

H. M. Becher,	Singapore Straits Settlements.
Talmadge Blass,	Brewster, N. Y.
Frederick W. Bradley,	Wardner, Idaho.
William Brown,	Phœnixville, Pa.
Louis E. Bryant,	Harrison, Tenn.
Robert E. Chambers,	New Glasgow, N. S.
Charles W. Chapman,	Melbourne, Australia.
John R. Claghorn,	Philadelphia, Pa.
Charles M. Conyngham,	Wilkes-Barre, Pa.
John N. Conyngham,	Wilkes-Barre, Pa.
Montgomery B. Corse,	Roanoke, Va.
E. L. Corthell,	Chicago, Ill.
Charles D. Cowland,	London, England.
George Davey,	Michoacan, Mexico.
William M. Davis,	Altoona, Pa.
Frank Eagle,	Chicago, Ill.
James J. Evans,	Cummings, Ill.
Emile E. Everett,	Iron Mountain, Mich.
Alberto Figari,	Lima, Peru.
P. H. Griffin,	Buffalo, N. Y.
Dr. F. Gildemeister,	Lima, Peru.
George Faunce,	Mansfield Valley, Pa.
John C. Haddock,	New York City.
J. S. Hartrick,	Melbourne, Australia.
George L. Heath,	South Lake Linden, Mich.
R. B. Hutchinson,	Denver, Colo.
John M. Higgins,	North Adelaide, South Australia.
George F. Hosking,	Bendigo, Otago, New Zealand.
Walter R. Ingalls,	New York City.
Orlando S. Johnson,	Scranton, Pa.
James F. Kemp,	New York City.
A. Koberle,	Bonne Terre, Mo.
John B. Law,	Pittston, Pa.
Andrew C. Lawson,	Berkeley, Cal.
William J. Love,	Embreeville, Tenn.
Burdett Loomis,	Hartford, Conn.
Richard McCulloch,	St. Louis, Mo.
David M. Meeker,	Newark, N. J.

Alfred T. Metcalf,	Johannesburg, South Africa.
I. C. H. Mingage,	Sidney, N. S. W.
Isaac R. Moister,	Wilkes-Barre, Pa.
J. A. Montgomery,	Birmingham, Ala.
Dr. Francisco de Monlevade,	Rio Janeiro, Brazil.
George F. Myers,	New York City.
Edgar C. Moxham,	Pulaski, Va.
George L. Norris,	Pencoyd, Pa.
Camilo E. Pani,	Mexico, Mexico.
Arthur C. Payne,	Orange, N. J.
W. G. Payne,	Kingston, Pa.
A. G. Phillips,	Johannesburg, South Africa.
J. D. Pope,	Phillipsburg, Mon.
Prof. James M. Porter,	Easton, Pa.
Rêne Proust,	Roxburg, Otago, New Zealand.
Rudolf Ruetschi,	Argentine, Kan.
Nathaniel Roberts,	Jersey City, N. J.
James P. Roe,	Pottstown, Pa.
Charles M. Schwab,	Braddock, Pa.
Harry P. Simpson,	Scranton, Pa.
Prof. Hjalmar Sjögren,	Upsala, Sweden.
Joseph Skelding,	Low Moor, Va.
E. J. Spencer,	Lynn, Mass.
John C. Slocum,	Chicago, Ill.
E. M. Touzeau,	London, England.
Robert S. Towne,	New York City.
W. H. Trewenack,	Sunny Corner, N. S. Wales.
Charles Vivian,	Scranton, Pa.
John R. Wagner,	Drifton, Pa.
Harry F. Walker,	Paterson, N. J.
Robert L. Walker,	Pittsburgh, Pa.
Thomas H. Watkins,	Scranton, Pa.
Samuel D. Warriner,	Wilkes-Barre, Pa.
William G. Wilkins,	Pittsburgh, Pa.
Morris Williams,	Shamokin, Pa.
Robert H. Wilson,	Saltsburg, Pa.

ASSOCIATES.

J. V. Darling,	Wilkes-Barre, Pa.
M. W. Evans,	New Orleans, La.
Albert Lewis,	Bear Creek, Pa.
Stephen J. Meeker,	Newark, N. J.
Chauncey G. Newton,	Buchtel, O.
E. E. R. Tratman,	New York City.

ASSOCIATES MADE MEMBERS.

Charles P. Coleman,
William S. Ackerman.

Excursions and Entertainments.

Thursday was spent in a delightful excursion through the Wyoming valley, by special train. The party was entertained at luncheon by Mr. Albert Lewis, at Harvey's Lake.

On Thursday evening a picturesque and charming "camp-fire" reception, given by General Paul A. Oliver, in the beautiful forest near his residence, was attended by hundreds of guests, who lingered until midnight among the unique attractions of the place.

Friday was spent in the Lehigh region, where the iron-breakers at Drifton, the great open-works of Coxe Brothers & Co. on the Mammoth vein, and other points of interest were visited. Luncheon was served in the woods at Oneida.

Minor excursions to Glen Onoko, the Mauch Chunk Switchback, etc., were arranged on Saturday for such as desired to remain.

Registry.

The following names were registered at the Glen Summit Hotel during the meeting. It is probable that many members, residing in the vicinity, did not register. The sessions and excursions were numerously attended by ladies accompanying members:

Robert Allison.
James Archbald.
James Archbald, Jr.
John Birkinbine.
T. Blass.
J. H. Bowden.
Jerome L. Boyer.
William McH. Boyer.
V. M. Braschi.
John A. Church.
R. N. Clark.
Charles M. Conyngham.
J. B. Coryell.
Eckley B. Coxe.
J. S. Cunningham.
W. B. Cogswell.
Edmund F. Davis.
David F. Day.
W. S. De Camp.
E. V. d'Invilliers.
Charles B. Dudley.
Daniel Edwards.
George H. Eldridge.
E. B. Ely.
John D. Evans.
John S. Fackenthal.
B. E. Fernow.
Frank Firmstone.
Rufus J. Foster.
John Fritz.
H. Stanley Goodwin.
Charles B. Green.
John C. Haddock.
John H. Harden.
G. C. Hewett.
Samuel Hines.
Thomas Hodgson.
W. H. Hoffman.
L. Holbrook.
J. F. Holloway.
W. S. Hungerford.
Alfred E. Hunt.
E. C. Huxley.
Baird Halberstadt.
W. R. Ingalls.
John H. Janeway, Jr.
Guy R. Johnson.
E. D. Johnston.
T. D. Jones.
Washington Jones.
M. S. Kemmerer.
William Kent.

C. Kirchhoff.
G. F. Knapp.
T. H. Knight.
D. W. Langdon, Jr.
N. M. Langdon.
W. A. Lathrop.
John B. Law.
Elmer H. Lawall.
R. G. E. Leckie.
A. C. Leisenring.
Walter Leisenring.
Albert Lewis.
A. McClellan.
Thomas L. McNair.
Robert Mannesmann.
John Markle.
Spencer Miller.
E. S. Moffat.
M. G. Moore.
S. F. Morris.
W. H. Morris.
F. H. Newell.
R. Van A. Norris.
Paul A. Oliver.
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E. M. Parrott.
S. B. Patterson.
W. G. Payne.
Richard Peters, Jr.
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J. C. Platt.
F. E. Platt.
Theodore D. Rand.
John C. F. Randolph.
R. W. Raymond.
Robert H. Richards.
H. S. Reets.
W. J. Richards.
William H. Richmond.
Axel Sahlin.
E. H. Sanborne.
H. B. Seaman.
H. J. Seaman.
A. W. Shaefer.
W. L. Shaefer.
William P. Shinn.
C. D. Simpson.
H. P. Simpson.
J. C. Smock.
H. C. Spaulding.
E. J. Spencer.
E. G. Spilsbury.
Irving A. Stearns.
H. H. Stoek.
A. H. Storrs.
S. Stutz.
Horace See.
W. T. Smith.
William Thaw, Jr.
H. G. Torrey.
John R. Wagner.
S. D. Warriner.
W. G. Welkins.
Oliver Williams.
Charles Vivian.
Leo Von Rosenberg.
C. H. Zehnder.

PAPERS.

THE CONSTRUCTION OF DETAILS FOR A MODERN LIXIVIATION-PLANT.

BY C. A. STETEFELDT, SAN FRANCISCO, CAL.

(Cleveland Meeting, June, 1891.)

A MODERN lixiviation-plant for the treatment of silver-ores with hyposulphite solutions differs so materially from its ancestors, that a critical description of the improvements recently carried out and proposed, will not be without interest to metallurgists. While the general arrangement of such a plant depends upon local circumstances, its details of construction are more or less constant and can be described from a general standpoint.

The plant consists of wooden tanks in which the ore is treated and solutions are accumulated; apparatus for elevating and transferring solutions, and for creating a vacuum below the filters of ore-tanks; filter-presses and drying-chambers for handling precipitates; apparatus for manufacturing sodium sulphide, etc.

§ 1. WOODEN TANKS.

Construction.—Tanks should be made of clear, well-seasoned lumber. In the United States, Oregon pine is the best material for this purpose. The staves, from 3 to 4 inches thick, according to size of tank, should be ordered cut to sweep of radius, and from 9 to 10 inches longer than the inside depth, but not "gained" for the bottom. The gaining of the staves, 1 inch deep, is done by hand, leaving a chine of 6 inches below the bottom. In all tanks the staves stand perpendicular to the bottoms. The bottom pieces, 3 to 4 inches thick, are cut to a diameter of 2 inches greater than that of the finished tank; they are grooved and joined by a tongue. All joints must be fitted with precision. White lead should never be put between the staves, but may be used in inserting the tongues between bottom pieces. The understructures of substantial timbers, placed on a solid foundation, should be sufficiently high to allow access to the bottom in case of leakage. The bottoms rest on joists, 3 to 4 inches wide and 10 to 12 inches deep, placed about 2 feet 6 inches apart, so that the staves are left entirely free. Hoops are made of

round iron, $\frac{7}{8}$ to $1\frac{1}{8}$ inches diameter, the threaded ends, with hexagonal nuts, passing through forged or cast-iron lugs, giving preference to the former. In order to get the full strength of the rods, the threaded ends are taken $\frac{1}{4}$ inch larger than the diameter of the rod. For tanks of large diameter, each hoop is made in two or three sections; this is necessary to effect a more uniform closing of the stave-joints by tightening the nuts in two or three places.

After finishing, the tanks are painted on the outside, staves and bottoms, with three coats of white lead.

Dimensions.—Formerly, the dimensions of lixiviation-tanks were taken quite small: Ore-tanks not larger than 12 feet diameter and 3 to 4 feet deep; precipitating-tanks, solution-sumps and storage-tanks of corresponding dimensions. In recent works, however, ore-tanks of 16 to 20 feet diameter and 8 to 9 feet depth; precipitating-tanks, solution-sumps and storage-tanks of 12 feet diameter and 8 to 9 feet depth are put up. As can readily be seen, the care and attention required to finish a charge in an ore-tank, or to precipitate a solution in a precipitating-tank, are independent of the size of the vessel; hence, the great advantages of large sizes.

The capacity of an ore-tank for twenty-four hours depends upon the specific gravity of the ore, the quantity of first and second wash-water, and of stock-solutions required for treatment, but principally upon the rate of lixiviation. Capacity increases in proportion to diameter, but remains nearly stationary so far as depth is concerned; that is, the same number of ore-tanks will be required whether their depth is 9 feet or only 4 or 5 feet, in order to treat a stipulated quantity of ore per day. In fact, should the rate of lixiviation increase with reduced depth, the same number of shallow tanks would put through in twenty-four hours more ore than deep ones. The principal advantage of increased depth consists, therefore, only in reducing the number of charges treated.

Sluicing Tailings.—Where water is abundant, tailings are now removed by sluicing, and great depth of the charge is no disadvantage. Even where water is scarce, and tailings have to be removed by hand, deep tanks should be used. It is only necessary to provide mechanical means for moving above the tanks large buckets into which the tailings are shovelled.

Filters for Ore-Tanks.—The false bottoms for the filter, and the latter itself, are prepared as follows: Wooden slats, $1\frac{1}{4}$ inches high and 1 inch wide, and separated 1 inch from each other, are fastened to the bottom. This has, so far, been done with iron screws, bedded

in white lead; I would suggest pins of hard wood. The inside of the slats, next to the bottom, is cut out in many places, $\frac{5}{8}$ inch deep and 3 inches wide, so that a free passage of the solution below the filter is established. Between the ends of the slats and the staves a clear space, $1\frac{1}{2}$ inches wide, is left. A strip of wood, $1\frac{1}{4}$ inches high and 1 inch wide, previously cut with a saw in many places, and well soaked in water, so that it will bend easily, is now fastened round the slats, leaving an annular space, $\frac{1}{2}$ inch wide, between the strip and the staves. One thickness of stiff matting, covering the slats and the circular strip, but not the annular space, forms the foundation of the filter-cloth proper. The latter, No. 10 canvas-duck, is cut to a diameter 6 inches greater than the inside of the tank, so that the ends can be pressed into the annular space described above, and kept in position by forcing down a $\frac{1}{2}$-inch rope.

Sluice-Gates.—A gate for sluicing tailings is illustrated in the *Trans.*, vol. xv., p. 359. The discharge-opening should be 18 to 20 inches wide and 8 to 9 inches high. The door is covered with a sheet of rubber, and should be suspended by a counter-weight when removed. For very large ore-tanks, say 18 to 20 feet diameter, it is desirable to have two sluice-gates diametrically opposite to each other. The bottom of the gate should be flush with the filter.

Solution-Outlets for the Ore-Tanks.—These are made of 2-inch, six-ply, rubber steam-hose, which is inserted in the following way: A piece of clear plank, 3 or 4 inches thick, is fastened to the bottom of the ore-tank with wood-screws, and a hole, having the exact size of the outside diameter of the hose, is bored through the bottom and the plank, at a flat angle of about 30°. Through this hole the hose is forced. All joints are made with thick white lead, the wood-screws being also bedded in this material. A pin of hard wood is finally driven through the end of the hose into the wood. If the solution-hose is permanently connected with a Montejus (see § 6) for producing a vacuum below the filter, and leaching with water from below the filter is desirable, a second hose should be inserted in the bottom of the tank and connected with the water-pipe. Finally, a third outlet may be provided and connected with a Koerting ejector or a geyser-pump (see § 7), if circulation of extra-solution is necessary.

Launders for Solutions in front of Ore-Tanks.—These launders, 6 inches wide and 8 inches deep, are made of clear $1\frac{1}{2}$-inch lumber and painted inside and outside with asphalt varnish. They are placed level, and should be held together with properly constructed

braces. Connection with precipitating-tanks is made by 3-inch, six-ply rubber hose, which is inserted in the same way as the hose in the bottom of the ore-tanks, but the hole is not bored slanting.

Launders are needed as follows: One for silver-bearing wash-water, one for waste water, one for solution, one for weak solution. If it is desirable to treat ore at different periods with cold and hot solutions, and to keep these solutions in rotation separately, a fifth launder must be added. Launders are either placed a short distance below the ore-tanks (from bottom of staves to top of launders, about 9 inches), and above the precipitating-tanks (about 6 inches from bottom of launders to top of precipitating-tank); or 5 to 6 feet above the top of precipitating-tanks, and below the upper-floor round ore-tanks. The first position is taken if no artificial means are used to increase the rate of lixiviation; the second if the rate of lixiviation is increased by a Montejus with vacuum. (See § 6.) The high position in the latter case is necessary to give sufficient head to the discharge through the 3-inch hose of the launder, when the solution is suddenly pressed up from the Montejus.

Boxes for Chemicals.—Solutions introduced to the ore-tanks are first conducted to a wooden box or barrel, with holes in the sides, standing on top of the charge. This prevents stirring up of the ore by a strong current of solution. These boxes also receive the copper sulphate and sodium hyposulphite for making extra-solution in the ore-tank.

Tail-Races.—If tailings are discharged by sluicing, proper attention should be paid to the construction of tail-races. Their inclination depends, of course, on the specific gravity and coarseness of the material; it should never be less than $\frac{3}{4}$ inch to the foot.

Precipitating-Tanks.

Stirrers.—For stirring by hand, an oar of ash, 16 feet long, is used. Mechanical stirrers are, however, far superior to hand-stirring. The best form of a stirrer is a propeller-screw of about 2 feet diameter, making 120 revolutions per minute. The screw revolves about 1 foot above the bottom of the tank. Since the lower end of the propeller-shaft can not be very well left without a guiding bearing, and at the same time should be protected against the corroding influence of the solution, it is enclosed in a heavy glass tube, held in position by rubber nipples and iron washers. The guiding bearing is made of *lignum vitæ*.

Stirring solutions with air, as described in the catalogues of the

Koerting Bros., is very effective. Some trials have recently been made with air-stirring at the Marsac mill; but it remains to be seen whether this method decomposes sodium hyposulphite in perceptible quantities or not.

Decanting-Pipes.—The decanting of the clear solution, after precipitation, is done by a swinging 2-inch gas-pipe, working through a stuffing-box. An illustration of this arrangement is found in the paper of Mr. Daggett (*Trans.*, xvi., 446). The pipe is marked there 4 inches, which is too large. When a propeller-stirrer is used, the pipe moving up and down cannot pass through the center of the tank, but must be shifted 18 inches away from it.

Outlet for Precipitates.—Precipitates are discharged by a 2-inch asbestos-packed angle-cock, inserted close to the bottom of the tank through the staves.

Launders.—In front of the precipitating-tanks are launders, leading the decanted solution to the solution-sumps, and running decanted wash-water or weak solution to waste. If cold and hot solutions are kept in rotation separately, two launders must be placed in front of the silver precipitating-tanks, one leading to the sump for cold, the other to that for hot solution. The launders at the back, conveying precipitates to the storage-tanks for sulphides, wash-water precipitate and lead carbonate, should be inclined, with a fall of about 4 feet in 100 feet. These launders are constructed in the same way as those in front of the ore-tanks, but need not be so large.

Storage-Tanks for Precipitates.

These are made about 10 feet in diameter and 3 feet deep. A 2-inch rubber hose forms the outlet for precipitates through the bottom. A 1-inch decanting-pipe for solution is inserted through the staves about 1 foot above the bottom. Both outlets are connected with the press-tank.

Solution-Sumps and Storage-Tanks.

These should be made of the same diameter as the precipitating-tanks, but about 1 foot less in depth. Two of each kind should be provided, especially where solutions are heated, and hot and cold solutions are kept in rotation separately.

§ 2. Precipitating-Tanks for Wash-Water.

If it is desirable to precipitate silver and copper from the wash-water by scrap-iron and sulphuric acid, the process is best conducted

in the apparatus described and illustrated in my book, *The Lixiviation of Silver-Ores*, page 159.

§ 3. Sodium-Sulphide Tanks.

The sodium-sulphide mixing-tank is made of cast-iron, 3 feet diameter and 7 feet deep. Its great depth is necessary because the concentrated lye foams considerably while adding the sulphur. The bottom is covered 1 inch deep with lead, to prevent its wearing out where steam is admitted for heating the lye. The two sodium-sulphide storage-tanks, receiving the diluted solution, are made of $\frac{3}{16}$-inch boiler-iron, about 6 feet in diameter and 5 feet deep. All the tanks are provided with 1½-inch asbestos-packed cocks. It may be desirable to increase the contents of sodium hyposulphite in the solution by rapid oxidation. For this purpose a gas-pipe coil, perforated by numerous small holes, is placed on the bottom of the tank and connected with an air-compressor. The air is then forced in small bubbles through the solution. Material and size of tanks for Solvay soda solution are not of importance.

§ 4. Apparatus for Heating the Stock-Solution.

Apparatus for heating stock-solution can be placed either in the solution-sumps or in the storage-tanks, depending on the most convenient location of the boilers. Coils of lead pipe, through which steam is conducted, are very effective, but, unfortunately, not sufficiently durable to warrant their cost. Although lead is not attacked by a pure hyposulphite solution, the stock-solution contains sulphates and chlorides, and these may form lead sulphate and chloride, both soluble in hyposulphite salts. Besides, we have to consider that the stock-solution is never entirely free from silver and copper.

At the Marsac mill, heavy cast-iron pipes one foot in diameter have been substituted for lead coils with satisfactory results regarding durability and cost.

I suggest giving to these heaters the shape of large, round, hollow disks, for a more economical utilization of steam.

Either heaters should be provided with steam-traps, or the escaping condensed water and steam should be conducted to the feed-water tank for the boilers.

§ 5. Apparatus for Raising Stock-Solution to the Storage-Tanks.

Plunger-Pumps.—Pumps of the Knowles type, as made and kept in stock by manufacturers, are not durable in contact with hypo-

sulphite solutions, none of the metals or their alloys resisting its corroding action. To make such pumps durable, valves, valve-seats and plungers should be made of hard rubber or glass. It is difficult to induce manufacturers to take such orders except at extravagant prices. The same objections apply to centrifugal and rotary pumps.

Pumps of the Koerting Type.—Such pumps must be made of material not affected by the solution. Their principal objection is the dilution of the solution by condensed steam.

The Geyser Pump.—If into a pipe, open at both ends, and immersed in a well, compressed air is introduced at the lower end, the solution in the pipe rises and is discharged, together with the air, at the upper end. The invention of this pump dates back several centuries, and it has recently received from Dr. Werner Siemens the appropriate name, "Geyser Pump." In the *Engineering and Mining Journal* of December 28, 1889, I have fully described it as Pohle's Air-Lift Pump, and, on the basis of the investigations of Messrs. Behr and Browne, of San Francisco, recommended its introduction in lixiviation-works. After I had made actual working-plans, however, for its introduction, some disagreeable features presented themselves, so that its application for raising stock-solution was abandoned. The objections consisted principally in the necessity of sinking a well, 30 feet deep, for the submersion-pipe, and in supporting a vertical solution-pipe 85 feet high.

The Montejus System.—A Montejus is an iron tank from which a solution is raised through a pipe by admitting steam or compressed air above the solution. (For protection of the inside, see § 13.) With steam, the tank should be in the shape of a long cylinder of relatively small diameter, standing upright, to avoid condensation as much as possible. With compressed air as motive power, the shape of the tank is not material. It is then best to place it so that the axis of the cylinder is horizontal. One of the heads of the cylinder is put on with bolts and a gasket, so that the interior can be easily coated with an asphaltic compound.

These tanks have two openings: one at the bottom, through which the tank is filled and discharged by a three-way cock, connected with the pipes leading to the solution-sump and the storage-tank; one on the top, through which air escapes while the tank is filling, and steam or compressed air is introduced to elevate the solution, which changes are also effected by a three-way cock.

For works of large capacity, two Montejus are placed side by side below the solution-sumps, so that one is filling while the other is dis-

charged. They should hold about 200 cubic feet. Convenient dimensions are: 4 feet diameter and 16 feet length, or 4½ feet diameter and 14 feet length, with solution-pipes from 3 to 4 inches diameter.

For large works, compressed air deserves decidedly the preference. The size of air-compressor, which should give about 30 pounds pressure per square inch, is the same for a geyser-pump as for a Montejus of equal capacity.

Although the effect of compressed air is better if applied to a geyser-pump than to a Montejus, the difference plays no important part in economy, while the original outlay for installation is in favor of the latter.

§ 6. Apparatus for Increasing the Rate of Lixiviation.

Suction-Hose.—The means hitherto employed for increasing the rate of lixiviation have been to give the hose of the solution-outlet below the filter of the ore-tank great length, and discharge the solution at considerable depth below the tank, taking care that the hose remains filled with solution, and creates a suction by the weight of the solution-column. This method is objectionable, not only because it is difficult to get rid of all the air below the filter, but also for the reason that the solution has to be raised again to the precipitating-tanks, unless a very steep mill-site permits their position much below the ore-tanks.

Pumps.—These are not recommended for reasons given in § 5.

Koerting Ejectors.—A Koerting ejector is effective provided it is not made of material liable to be corroded by the solution. The lead-lined ejectors, sold by A. Allen, New York, are not durable; even the platinum steam-nozzles wear out. Ejectors of porcelain, with hard rubber steam-nozzles, are manufactured by the Koerting Brothers, Hanover, Germany. The great objection to a Koerting ejector is, however, the dilution of the solution by condensed steam. The apparatus also acts intermittently, unless carefully regulated to meet the capacity of the filter.

Montejus.—The best effect is unquestionably produced by a Montejus. For this purpose the opening at the bottom is connected with a pipe through which, by a three-way cock, the solution either runs in from the hose at the bottom of the ore-tank, or is raised to the launders in communication with the precipitating-tanks. The outlet on top has a pipe with a three-way cock, through which compressed air can be introduced for raising the solution, or a suction is created

by a Koerting vacuum-pump for filling the tank, or communication is shut off both from the air-compressor and vacuum-pump, provided the apparatus is idle. Only one Koerting vacuum-pump, producing a vacuum of 68 cm. quicksilver, is needed for a number of Montejus tanks, the latter being all in communication with a pipe, at the end of which the Koerting is placed.

These vacuum-pumps are best obtained from the Koerting Brothers, Hanover, Germany.

Necessarily, the operation of the apparatus is intermittent; but the discharge of the Montejus takes such a short time that this is not detrimental.

§ 7. Apparatus for Circulating Extra-Solution.

In circulating extra-solution of standard composition, only a quantity is used sufficient to saturate the charge in the ore-tank, and this is allowed to filter repeatedly and continuously through the ore. As will readily be seen, the Montejus system cannot be used for this purpose. Here we must rely either upon a Koerting ejector, made of porcelain, or use the geyser-pump. Which of the two deserves the preference is doubtful, the geyser-pump being untried. Their respective advantages and disadvantages may be stated as follows:

With the Koerting there is risk of decomposing extra-solution with formation of sulphates and Cu_2S, the solution coming in contact with the hot steam, and of weakening the solûtion by dilution. The heating, however, is beneficial. Circulation is also quicker by material increase of the rate of lixiviation.

In the geyser-pump, the extra-solution comes in contact with air only, whereby it may deteriorate through the conversion of the sodium hyposulphite into tetrathionate, but without affecting the cuprous hyposulphite. By this reaction, and by the evaporation of water, the extra-solution will gain in concentration. On the other hand, the solution will be reduced in temperature and circulation will be slower, because the rate of lixiviation will not be materially increased, even by making the submersion-pipe as long as practicable. The original cost of installation, and the expense of running, would hardly enter into calculation. Only an actual comparative test can decide which apparatus is the better.

§ 8. Filter-Press and Press-Tank.

For lixiviation-mills of various capacities I recommend Johnson's filter-press, with 24 to 36 chambers, 15 to 18 inches diameter, with-

out distance-rings and with closed delivery. Closed delivery obviates the necessity of having a sump below the press, the solution being at once raised to the main solution-sump.

If it is desirable to free the precipitates from adhering stock-solution, for reasons given in my paper on "Refining of Sulphides" (p. 37 of this volume), a filter-press should be used in which the cakes can be washed after the chambers have been filled.

The chambers are of cast-iron, with a coating of asphalt varnish, which should be occasionally renewed.

The proper way of charging the filter-press is by a press-tank. In Mr. Daggett's paper (*Trans.*, xvi., 438), a press-tank is illustrated and described, constructed for steam-pressure. For works of large capacity, steam should be replaced by compressed air. In the latter case the wooden diaphragm and the pipe extending to the bottom are superfluous, and the precipitates are charged through an opening in the side of the tank near its top.

The construction, as shown in the cut, is not recommended. The bottom as well as the top should be made of boiler-iron or steel-sheet, not of cast-iron. At the Marsac mill, one of these tanks exploded, by breakage of the cast-iron bottom, at a pressure of 80 pounds. For compressed air the tank should be constructed to be safe for 150 pounds pressure to the square inch.

§ 9. Drying-Chamber for Precipitates.

A drying-chamber for sulphides and other precipitates, is illustrated in the paper just cited, pp. 440 and 443. Its construction is contrary to all rules for a good system of drying by hot air. A chamber of this kind, in use at the Marsac mill, has a very low efficiency. Correct principles of construction are to heat the air by a coil of steam-pipes in a separate chamber attached to the dryer, to allow the heated air to enter the drying-chamber at the top, and circulate downward between the iron pans charged with precipitates, and finally to withdraw the moist air at the bottom by a Koerting steam-jet ventilator. Upon these principles all modern apparatus for desiccation is constructed.

The outlet of the steam-pipe may be either connected with a steam-trap or with the mud-drum of the boiler. The latter can only be done if the boiler is so located that its mud-drum is lower than the outlet of the coil. In this case the coil works on the gravity principle, the most economical method of utilizing the heat of the steam. The coil should be tested for leakage at a pressure of 80 pounds.

§ 10. Auxiliary Machinery.

Air-Compressors.—For Montejus or geyser-pumps and for the press-tank, two separate air-compressors are needed, the one furnishing air of about 30 pounds, the other of 150 pounds pressure. The size of the compressors must be calculated to meet the demands of the capacity of the works. The Montejus air-compressor should have a capacity to raise all the stock-solution needed per day in about 8 hours, so that the plant is not in operation all the time and the solution can be transferred quickly.

Sulphide-Sampler.—The breaking-up of the lumps of sulphides and other precipitates, as they come from the drying-chamber, is done in an ordinary small sample-grinder, as sold by any western foundry.

Boilers.—If considerable steam is needed for heating the stock-solution and wash-water prior to its precipitation by scrap-iron; also for running Koerting ejectors and vacuum-pumps, a separate boiler is desirable for the lixiviation-plant proper.

Fire-Pumps for Sluicing Tailings.—If the water in the pipe-line has not sufficient head for sluicing tailings, a fire-pump for this purpose becomes necessary. A steam-pump of the Knowles type may be used with 10-inch steam- and 5-inch water-cylinder, 12-inch stroke and 4-inch suction and discharge. But I would recommend a Koerting steam-jet fire-pump, which is much cheaper than a plunger-pump and not less efficient.

§ 11. Pipe-Lines.

Iron gas-pipes are used for conveying stock-solution and sodium sulphide solution.

They should be joined by flange-couplings. The size of the stock-solution-pipes above the ore-tanks depends on the capacity of the works, and the position of the storage-tanks, producing more or less head. In large works they are generally taken from 3 to 4 inches diameter.

§ 12. Cocks.

The only cocks resisting for a long time the corroding action of lixiviation-solutions are the Pratt-Cady iron-body, Bower-Barffed plugs, asbestos-packed cocks, sold by Fairbanks & Co., New York. These cocks last very well, even on the sodium-sulphide mixing-tank. They were first introduced in lixiviation-works by the writer at the Marsac Mill, Park City, Utah.

§ 13. Protection of Iron by Asphaltic Covering, and other Notes on Construction.

Castings and Pipes.—Castings and pipes exposed to hyposulphite solutions must be protected by an asphaltic covering; the pipes, of course, on the inside. Pipes conveying sodium sulphide solution do not require such covering. A pure asphalt, called "Maltha," manufactured in San Francisco, the residue from distillation of heavy California mineral oils, now shipped in considerable quantities to the eastern States, is exceedingly well adapted to this purpose. The maltha is melted in a sheet-iron trough, and the clean and dry castings or pipes are immersed in the bath. They should remain there until the iron has reached the temperature of the liquid asphalt; this insures firm adhesion of the asphaltic covering to the iron. The propeller-screws and shafts for the precipitating-tanks are covered in the same way.

Montejus tanks and the press-tanks need different treatment. Here a malthine varnish is used. To the clean and dry inside of the vessels, a coat of varnish is applied. After drying, a second coat is put on, and, as the painting progresses, cotton cloth—weight about 6 ounces per yard—is pasted on, care being taken to saturate overlapping joints with varnish. On top of the tank the cloth is drawn over the flange. The cover is treated in the same way. Finally, a third coat of varnish is put over the cotton cloth. After perfect drying, the cover is bolted on, using a soft rope saturated with varnish for a gasket.

Fastening Castings and Flanges to Tanks by Bolts.—Bore holes to fit bolts closely; countersink bolt-heads into wood and cover with thick white lead. Washers behind bolt-heads are also bedded in white lead. Bolts are first covered with asphalt.

Connecting Rubber Hose with Gas-Pipe Nipples.—The following method was communicated to me by F. M. Johnson: Draw out the nipple to fit the inside of the hose closely; heat the nipple so that it will melt rubber without burning it; then press the nipple into the hose about 4 inches and immerse immediately in asphalt varnish for cooling. The hose adheres so firmly to the iron that it can only be removed by cutting.

THE PRECIPITATION OF METALS FROM HYPOSULPHITE SOLUTIONS.

BY C. A. STETEFELDT, SAN FRANCISCO, CAL.

(Cleveland Meeting, June, 1891.)

METALLURGICAL processes cannot be conducted successfully without the aid of analytical chemistry. The great perfection of lead-smelting in the West, for instance, has only been accomplished by the analysis of ores, fluxes, slags and all products of the furnace. Ores are mixed and fluxed to obtain a slag of desired composition. Account is even taken of the composition of the ashes obtained from the coke. The "muscular" smelter has left the field forever! This state of affairs has been induced by sharp competition, *i.e.*, by a complete separation of the industries of mining and reducing ores, not by the liberality and wisdom of directors and stockholders to provide laboratories and engage chemists, or by their love for scientific investigation.

It would be considered absurd, at present, to run a lead-smelter, a blast-furnace for pig-iron, or a Bessemer plant without the assistance of a well-equipped laboratory and a chemist; but it is considered quite sufficient to provide amalgamation- or lixiviation-works merely with a crude assay-office, and an assayer who is paid less than a laborer in the mill.

In my opinion, this will change in the near future. By the exclusion of Mexican lead-ores, and their growing scarcity in the West, smelters have been forced to raise their charges on so-called dry silver-ores, and cannot handle the latter now in such unlimited quantities as formerly. The inevitable result must be that the surplus of these dry silver-ores, especially those of low grade, will be reduced by processes cheaper than smelting; and here the introduction of lixiviation has an excellent field, provided the muscular lixiviator has an able chemist for an assistant.

It is to be regretted that so little analytical work has been done in lixiviation. We know almost nothing of the composition of roasted ores before and after lixiviation; of the constitution of the first wash-water and of lixiviation-solutions after prolonged use; of the chemistry of sodium and calcium sulphides; of the composition

of sulphide and carbonate precipitates. Without such work material progress is not possible.

During the last seven years a great number of papers on lixiviation have been published, containing a good deal of valuable information, but also many errors due to hasty and incomplete investigation without thorough analytical work. Writers of metallurgical treatises, myself included, have repeated these errors like parrots—what else could they do! I will point out a few. We find it almost everywhere stated that in boiling caustic milk of lime with sulphur the lower calcium polysulphides cannot be produced, because they are insoluble. This is not correct. CaS cannot be obtained in the wet way or in solution. If it is made by heating $CaCO_3$ in fumes of CS_2 and CO_2, and brought in contact with water, CaH_2O_2 and CaH_2S_2 are formed, the latter being readily dissolved. According to the best authorities, CaS_2 and CaS_3 do not seem to exist at all, at least not in aqueous solution. If caustic lime is boiled with sulphur, the tendency prevails to form CaS_4 even if an excess of lime is present, not because CaS_2 and CaS_3 are insoluble compounds, but because calcium has stronger affinities to unite with four equivalents of sulphur.

A more serious error occurs in Daggett's paper, *Trans.*, xvi., 428, in regard to the abnormally high precipitating-coefficients of sodium sulphide prepared according to Russell's directions. Believing in the correctness of Russell's experiments, I built upon them an elaborate theory in my book on the *Lixiviation of Silver-Ores with Hyposulphite Solutions.* Aaron was the first who proved that these abnormal precipitating-coefficients exist only in partial precipitation, which Russell had entirely overlooked; hence, they have no existence in practical mill-work. That Aaron's results are correct, I know from personal observation. In the following I have to record another important error. In Daggett's paper, already quoted, we find on page 430 that calcium monosulphide is precipitated by Russell's sodium sulphide if the hyposulphite solution contains calcium salts. The yellow precipitate of calcium sulphide is said to appear after nearly all the silver, copper and lead have fallen out. I have no doubt the precipitate occurs, and looks yellow; but it is not CaS, or CaS_2, or any other calcium sulphide. Like all other faithful writers of books, I have also endorsed this statement in my treatise on lixiviation.

The metals playing an important part in precipitation from hyposulphite solutions, resulting from the lixiviation of silver-ores, are

silver, copper, lead, and calcium. The quantity of gold that may be present is too small to enter into calculation. Silver and copper are always precipitated as sulphides by sodium or calcium sulphide; lead and calcium may be precipitated as carbonates by Solvay soda; or lead alone as hydroxide by caustic lime. Sodium sulphide can be used either as a mixture of Na_2S and NaS_2, which we will designate in future as "Russell's Sulphide," or NaS_2, or a higher poly-sulphide. Calcium sulphide is always used as CaS_5. For precipitating lead, caustic milk of lime is prepared with hyposulphite stock-solution. Solvay soda is also dissolved in stock-solution.

Calcium may also be precipitated by a sodium sulphide solution, as will be fully discussed later on. Caustic lime and Solvay soda are always used before calcium or sodium sulphide.

In the absence of calcium, the lead carbonate precipitated by Solvay soda can be obtained practically free from silver, that is to say after washing it with fresh hyposulphite solution. From Ontario ore, for instance, Russell obtained in this way lead carbonate with only two ounces of silver per ton. If calcium is present, the lead-calcium carbonate contains more or less silver, and may be very rich. According to Wilson and Russell, the silver does not increase in proportion to the calcium precipitated. Russell also claims that washing the precipitate with pure or more concentrated hyposulphite solution will reduce its value in silver considerably.

This is not clearly established. At the Marsac mill, one lot of carbonates was recently produced containing 8.1 per cent. lead, and 392 ounces silver per ton; and another lot containing 19 per cent. lead, and 1269 ounces silver per ton. I do not know the exact circumstances under which these carbonates were obtained. Evidently, this subject, like so many others in lixiviation, needs a thorough analytical investigation.

Lead hydroxide precipitated by caustic lime is generally rich in silver. According to Rueger, the lead precipitate obtained at the Mount Corry mill, Nevada, contained 60 per cent lead and 420 ounces and more of silver per ton.

The separate precipitation of lead and calcium may be not only economical directly, but also indirectly by the production of silver sulphides of higher grade.

Since the introduction, by Kiss, of CaS_5 as precipitant, the majority of metallurgists have tenaciously adhered to this practice, evidently without examining the subject critically. The question has become more complicated by the recent introduction of methods for separate

precipitation of lead, and of the Russell process. It is hardly necessary to add that the problem also includes the question whether a sodium or calcium hyposulphite solution deserves preference in lixiviation.

Although lixiviation is now always commenced with sodium hyposulphite, this salt is gradually replaced by calcium hyposulphite if CaS_5 is used as precipitant.

In the following I propose to present in a systematic manner the most important facts that have a practical bearing upon the subject of precipitation. The reader, however, should not expect a complete and exhaustive treatise, for which analytical data are entirely lacking.

§ 1. Preparation of Sodium and Calcium Sulphides.

Both reagents are prepared by boiling either a concentrated lye of caustic soda, or caustic milk of lime with sulphur. In making calcium sulphide, sufficient sulphur must be added to the caustic lime to obtain CaS_4, because lower calcium sulphides are not formed. In fact, the solution contains CaS_4 even if an excess of lime is present, while an excess of sulphur produces CaS_5. From the slight solubility of calcium hydrate in water it follows that this process must require considerable time, and that at the end of the operation, a solution of only moderate concentration must result. From sodium hydrate, on the contrary, a lye of almost any concentration can be obtained, as well as sodium monosulphide, and all of the polysulphides. Hence, the process must be completed very rapidly, and, if the lower polysulphides are desired, with a minimum consumption of sulphur.

The mode of preparing calcium sulphide is well known. I will only call attention to the facts that it is troublesome, wasteful in chemicals, amount of steam and time consumed, and that an excess of sulphur should be used to obtain CaS_5.

Russell discovered a practical and easy method of preparing sodium sulphide. According to his directions, the contents of a drum of caustic soda are broken into lumps and dissolved with the aid of steam in a minimum of water, thus forming a very concentrated lye. The operation is conducted in a cast-iron tank 3 feet diameter and 7 feet high. When the lye has reached a temperature of not less than 100° C. sulphur is gradually added, two-thirds of the weight of the caustic soda. A most violent reaction takes place, the mass foaming considerably. This, however, is caused by disen-

gagement of steam and not of any other gas. After all the sulphur has disappeared, the product is allowed to cool, dissolved by adding hyposulphite stock-solution, and discharged into a storage-tank, where the solution can be diluted still further. In two hours, 700 pounds of caustic soda, the contents of one drum, can be easily converted into sodium sulphide. The solution contains a mixture of Na_2S and Na_2S_2.

The quantity of sulphur used for making Russell's sulphide is somewhat empirical. The two-thirds rule may be very convenient for the laborer; but metallurgy has not been invented for his convenience. As will be seen at once, the rule does not at all consider different grades of caustic soda, which may contain from 85 to 95 per cent. NaHO, thus producing sodium sulphide differing materially in composition. If we use sulphur with 95 per cent. S, and caustic soda with 95 per cent. NaHO, the resulting sulphide will be $Na_2S + Na_2S_2$. If, however, the caustic soda contains only 85 per cent. NaHO, the sulphide will have very nearly the composition $Na_2S + 4\ Na_2S_2$.

This subject will be further discussed in the following paragraphs:

§ 2. Formation of Hyposulphite Salts.

The reactions taking place in boiling caustic soda with sulphur are expressed by the equation:

$$(1.)\quad 6\ NaHO + (2 + 2x)S = 2\ Na_2S_x + Na_2S_2O_3 + 3\ H_2O.$$

That is to say, for two equivalents of a sodium sulphide, one equivalent of the hyposulphite salt is formed.

For the formation of different sodium polysulphides, the quantities of sulphur consumed would be as follows:

100 c. p. Caustic Soda require c. p. Sulphur.	100 Caustic Soda, with 90 per cent. NaHO, require Sulphur with 95 per cent. S.	For the formation of
53.3	50.5	Na_2S.
80.0	75.8	Na_2S_2.
106.6	101.0	Na_2S_3.
133.3	126.3	Na_2S_4.
160.0	151.6	Na_2S_5.

In manufacturing sodium sulphide there are no very important side-issues to the reactions expressed by equation No. 1; and, taking into consideration the impurities of commercial caustic soda and sulphur, calculations for practical purposes can be based upon them. There are, however, certain limitations to the equation. It is not true, for instance, when $x = 1$. In using an amount of sulphur only sufficient for the formation of Na_2S, caustic soda remains free, and polysulphides are formed besides Na_2S. It is even difficult to obtain a reagent absolutely free from caustic soda by using enough sulphur to form Na_2S_2. Such a solution is also liable to contain some Na_2S and polysulphides higher than Na_2S_2, a tendency prevailing to form the higher polysulphides. If we replace in equation No. 1 sodium by calcium, the reactions become still more complicated. In the first place, the equation is only true for CaS_4 and CaS_5. It is very probable, however, that some sulphydrates are also formed, and this seems the case even in preparing sodium sulphide. I will here draw attention to the fact that H_2S frequently escapes at the end of precipitation, even if the solution has originally an alkaline reaction. This is the case in using both calcium and sodium sulphide as precipitants. The formation of H_2S can be easily explained if we assume the existence of sulphydrates. The reaction would be as follows: $Ag_2S_2O_3 + CaH_2S_2 = Ag_2S + CaS_2O_3 + H_2S$. Naturally, the H_2S only escapes when precipitation of the metals is nearly completed.

This also explains an observation made by Russell at the Marsac mill.

He says: "The solution running from the ore-vats to the precipitating-tanks is perfectly neutral, but after precipitation with sodium sulphide it is so acid that blue litmus-paper is instantly reddened by it. At the end of precipitation the odor of H_2S is very perceptible; it even appears during precipitation if stirring is not well done."

This, it seems to me, shows the presence of sulphydrates. The H_2S formed in the beginning of precipitation is absorbed by the solution, throwing down metals and setting acid free. To return to the calcium sulphide:

The reaction is also disturbed by the formation of oxy-sulphurets —which are not easily soluble—especially if sulphur is not used in excess to form CaS_5, and by decomposition of calcium hyposulphite at boiling-point. While a sodium hyposulphite solution can be

heated to boiling without detriment, that of the calcium salt decomposes above 60° C. as follows:

$$CaS_2O_3 = CaSO_3 + S.$$

This decomposition is, however, resisted in the presence of CaS_4 and CaS_5, and counteracted by the opposite reaction:

$$CaSO_3 + S = CaS_2O_3,$$

which takes place between 30° and 40° C., that is to say, after cooling and settling of the solution. To what extent these opposing reactions balance each other is not known, and depends, most likely, upon various circumstances.

For these reasons it is not possible to calculate beforehand, even approximately, from the chemicals consumed, the effect of a calcium sulphide solution in precipitation, or to compare it, merely by calculation, with the same effect of a sodium sulphide solution. Only actual mill-statistics make a comparison between the two reagents possible. It can readily be seen, however, that the preparation of CaS_5 must involve a waste of chemicals compared with that of Na_2S_2.

According to equation No. 1, 100 parts of caustic soda, consumed in the manufacture of sodium sulphide, will produce 103.3 parts of $Na_2S_2O_3 + 5$ aq.

Or 100 parts commercial caustic soda, containing 90 per cent. NaHO, would produce 93 parts $Na_2S_2O_3 + 5$ aq. Both sodium and calcium sulphide solutions, if exposed to the air, oxidize with formation of hyposulphite salts, but the latter more rapidly than the former.

In case Na_2S is exposed to the atmosphere, sodium hydrate is formed besides hyposulphite:

$$(2.) \qquad 2\,Na_2S + H_2O + 4O = Na_2S_2O_3 + 2NaHO.$$

The caustic soda is then converted into carbonate by absorbing carbonic acid from the air. Na_2S_2 is completely converted into hyposulphite, while the higher polysulphides are oxidized with precipitation of free sulphur.

O. Hoffman, who used CaS_5 as precipitant at the Silver King mill, Arizona, stated to me that the original stock-solution was used over a year and a half, and that it increased in strength and volume, making it necessary to run a part of it to waste.

In this case a large amount of copper and lead was precipitated

with the silver. This is good evidence to show how rapidly CaS_5 oxidizes. What has been said about the more rapid decomposition of calcium sulphide in contact with air, compared with sodium sulphide, holds good regarding the respective hyposulphite salts.

Russell exposed solutions of different concentration in soup-plates to the atmosphere for seven days, at a temperature of from 20° to 22° C., and brought them to their original volume at the expiration of that time. The relation of the depth of the solution to the diameter of the plates was about 1 to 8. The deterioration which had taken place was for sodium hyposulphite, 1.4 per cent., and for the calcium salt, 16.1 per cent.

Formerly lixiviation-solutions were never heated; the new practice finds it profitable to operate with solutions of a temperature as high as 50° C. Would not the loss in calcium hyposulphite be much increased in such a case, when complete decomposition takes place above 60° C? The stubborn advocates of the old practice in lixiviation say: We never needed to buy sodium hyposulphite; our stock-solution increased in strength, and we had to run it finally to waste! But at what expense, gentlemen, in caustic lime and sulphur, and inferior extraction in silver, did you reach this result?

§ 3. Economy of Producing Hyposulphites by Oxidation of Sulphide Solutions.

The question is of importance, whether it is cheaper to buy and add sodium hyposulphite to a deteriorated stock-solution, or to introduce hyposulphite by allowing the sulphide solution to oxidize, whereby it degenerates in precipitating power, but gains in hyposulphite. This depends upon local prices of chemicals. In most cases it will be cheaper to produce the hyposulphite by oxidization of the sulphide. A calculation under assumed conditions will be of interest. For Na_2S_2 we find:

The formation of 100 pounds $Na_2S_2O_3 + 5aq$ requires:

35.8 pounds caustic soda of 90 per cent., at 5.5 cts.	$1.96
27.2 pounds sulphur of 95 per cent., at 3.5 cts.	0.95
Total cost	$2.91
100 pounds $Na_2S_2O_3 + 5aq$, at 4 cts.	4.00
Leaving a saving of	$1.09

if this salt is obtained by the oxidation of Na_2S_2 in the mill.

The cost of obtaining calcium hyposulphite from the sulphide

solution by oxidation cannot be accurately or even approximately calculated, for reasons already stated. From the fact that CaS_5 must be oxidized in place of Na_2S_2, and that in the preparation of calcium sulphide relatively more chemicals are consumed, it follows that the cost cannot, in most cases, be materially less, although caustic lime is very much cheaper than caustic soda; and it may be even greater.

To use a calcium sulphide solution so highly oxidized that the stock-solution increases in strength and volume, making it necessary to run it to waste, cannot be good economy, because it necessitates an excessive consumption of the precipitant.

Should it be desirable to use an oxidized sodium sulphide solution, containing a large amount of hyposulphite, oxidation is most conveniently and quickly effected by forcing air through a coil of gas-pipe, provided with a great number of small holes, and placed at the bottom of the storage-tank.

§ 4. Precipitation of Calcium by Sodium Sulphide Solutions.

This subject has never been considered or presented before. I have observed in former paragraphs:

1st. That it is difficult to prepare the lower sodium polysulphides absolutely free from caustic soda.

2d. That if a sodium sulphide solution containing Na_2S is exposed to the atmosphere, $Na_2S_2O_3+2NaHO$ are formed.

3d. That free caustic soda absorbs carbonic acid from the atmosphere and becomes Na_2CO_3.

In these reactions lies the key to the appearance of the mysterious yellow precipitate which Russell took for CaS. As can readily be seen, if Russell's sulphide is for any length of time exposed to the atmosphere, it must contain more or less Na_2CO_3. The Na_2CO_3 originally contained in commercial caustic soda—which may vary between 3 and 5 per cent.—also comes into play. Assuming, for instance, that 6 pounds caustic soda were consumed (manufactured into sodium sulphide) per ton of ore, containing 4 per cent. Na_2CO_3, there would come into action 0.24 pound Na_2CO_3, precipitating 0.09 pound $Ca=0.22$ pound $CaCO_3$. Assuming that Russell's sulphide had been made of a high-grade caustic soda (with 95 per cent. NaHO), having the composition $Na_2S+Na_2S_2$, and that by long exposure to the air, one-half of the Na_2S became oxidized, how much Na_2CO_3 would a quantity of Russell's sulphide contain

produced from 6 pounds caustic soda? Exactly 1.28 pounds, precipitating 0.48 pound Ca = 1.20 pounds $CaCO_3$. Adding the sodium carbonate originally present in the caustic soda, the total amount would be 1.52 pounds, precipitating 0.57 pound Ca = 1.42 pounds $CaCO_3$.*

A sodium sulphide solution containing Na_2S, if allowed to oxidize, is unprofitable for another reason. Equation No. 2 shows that only one-half of the Na_2S is converted into $Na_2S_2O_3$, the other half becoming NaHO. Thus oxidation of the solution is unsatisfactory in two ways.

If Russell's sulphide is to be used to advantage, it should be preserved in hermetically sealed tanks.

It has been shown in § 3 that sodium hyposulphite is obtained in most cases at less expense by allowing the sodium sulphide solution to oxidize than by buying the hypo-salt; hence, it seems timely to abolish Russell's sulphide altogether and substitute in its place Na_2S_2, or even a higher polysulphide, if Na_2S_2 cannot be obtained free from Na_2S.

All the statements made above are supported by actual experiments. I prepared from chemically pure reagents two sodium sulphide solutions, one with sufficient sulphur to contain $Na_2S + Na_2S_2$, another to contain Na_2S_2. As I am fully aware, it is very difficult to prepare such solutions on a very small laboratory-scale, free from caustic soda or from higher polysulphides; but this does not influence in the least the principal points at issue.

The following tests were made with the freshly prepared sulphide solutions:

First, solutions of calcium chloride and of gypsum in sodium hyposulphite were prepared, of such concentration that caustic soda produced in them a precipitate of calcium hydrate.

To these solutions the sodium sulphides were now added. Each drop of the concentrated reagents produced a slight flocculent pre-

* I have received from the Marsac mill some data concerning silver precipitated from wash-water and weak solution, and from the regular lixiviation-solution; also, the values of these precipitates in silver and their percentages of lead and copper. According to these figures, the consumption, per ton of ore, of caustic soda in the manufacture of Russell's sulphide should be much less than it actually is, unless it be assumed that more than one-half the Na_2S in Russell's sulphide is oxidised by contact with air, a supposition with which the figures agree remarkably well. I do not give the details of this calculation, because I do not know whether the values on which it is based represent true averages.

cipitate of intensely yellow color, which would disappear again upon stirring. After successive additions of the reagents, the precipitates finally remained, but to effect this, much more of the solution containing $Na_2S + Na_2S_2$ was consumed than of that containing Na_2S_2. It became at once apparent that only a very small portion of the reagents was engaged in the reaction, the solution assuming an intensely yellow color. Upon adding a sufficient quantity of water the precipitates dissolved again; hence, it was not possible to examine them further by filtering and washing with water. This was done, however, with alcohol, in which sodium polysulphides are soluble and calcium hydrate is not. After repeated washing with alcohol the precipitates became white and proved to be calcium hydrate.

Now, 50 c.c. of each sulphide solution was put into a 7-pound acid bottle and shaken for three hours, air and carbonic acid being blown from the lungs into the bottles every few minutes. In the beginning, the solutions remained perfectly clear; finally, a small amount of sulphur separated, indicating the presence of higher polysulphides than Na_2S_2.

The oxidized sulphides were now tested with calcium solutions as before. Yellow precipitates appeared at once; but with equal quantities of the reagents, the precipitate from the Na_2S_2 solution was quite small compared with that obtained by the $Na_2S + Na_2S_2$ solution. These precipitates were not flocculent, settled quickly and did not dissolve upon addition of water. After washing they became perfectly white, and proved to be $CaCO_3$. Q. E. D!

It is claimed that the precipitate of calcium can be avoided if precipitation of silver, copper and lead is not done too closely. I am of an entirely different opinion. If the precipitate were actually an insoluble calcium sulphide, yes; but since it is $CaCO_3$, its precipitation must commence at once. The precipitate becomes apparent only after the metals have been completely precipitated and if calcium still remains in solution. The precipitate appears yellow, because it holds mechanically sodium sulphide. Only prolonged washing makes it white. As will be seen from the example quoted previously, much more calcium may be in a lixiviation-solution per ton of ore than can be precipitated even by a well oxidized Russell's sulphide prepared from 6 pounds of caustic soda.

It remains to be seen how much sulphur is actually needed in practice to prepare a sodium sulphide free from Na_2S. How to test such a solution for ascertaining this, has been shown above.

§ 5. Regeneration of Sodium Hyposulphite from Tetrathionate.

In preparing Russell's extra-solution, one pound of sodium hyposulphite is changed to tetrathionate for each pound of copper sulphate consumed. The formation of tetrathionate is further increased by the atmospheric decomposition of extra-solution, so that, finally, the solution must hold a considerable amount of sodium tetrathionate, which salt is not a solvent for silver-compounds. Fortunately, a very simple reaction converts the tetrathionate again into hyposulphite:

$$Na_2S_4O_6 + Na_2S_x = 2Na_2S_2O_3 + xS.$$

This reaction takes place in precipitating the solution.

So long as metals remain unprecipitated, it is doubtful whether this regeneration can take place or not. Close precipitation and over-precipitation, however, would give to the reaction free scope. Hence, it will be judicious to precipitate closely, and to over-precipitate occasionally, the solution in a number of precipitating-tanks. The excess of sodium sulphide is then neutralized by running some fresh silver solution into the precipitating-tank. That all this is not merely theoretical speculation, is proved by statistics from the Yedras mill, Mexico. With a consumption of 9.6 pounds copper sulphate per ton of ore for making extra-solution, the loss in sodium hyposulphite, by close precipitation, was only 1.4 pounds per ton of ore treated. When the consumption of copper sulphate was reduced to 7.7 pounds, the loss in hyposulphite fell to 0.74 pound. In this case over-precipitation was not practiced; if it had been done, the loss in hyposulphite would have been reduced still further. That it is profitable to regenerate hyposulphite from tetrathionate, the following calculation will show, based upon the quality and prices of chemicals assumed in § 3.

For the production of 100 pounds sodium hyposulphite, a quantity of Na_2S_x would be needed, requiring:

21.5 pounds caustic soda,	\$1.18
16.3 pounds sulphur,	0.57
Total cost,	\$1.75
100 pounds $Na_2S_2O_3 + 5$ aq,	\$4.00
Net saving,	\$2.25

In this estimate the hyposulphite contained in and added by freshly

prepared Na_2S_2 is included. In regenerating hyposulphite from tetrathionate, free sulphur is added to the precipitated sulphides, as follows:

For 100 parts hyposulphite regenerated, there are added:

When Na_2S is used,	. . . . .	6.45 parts of sulphur.
" Na_2S_2 "	. . . . .	12.90 " "
" CaS_5 "	. . . . .	32.25 " "

In these figures the hyposulphite added by and contained in the sulphides is not included.

§ 6. Precipitating-Coefficients.

The precipitating-coefficients of a sulphide solution are the quantities of silver, copper and lead precipitated for 100 parts of caustic soda, or caustic lime consumed in its manufacture. In the same way precipitating-coefficients can be established for sulphur.

If a metal is precipitated by an alkaline polysulphide, RS_x, one equivalent of the latter precipitates one equivalent, or a double equivalent, of the former, $(x-1)S$ being liberated as free sulphur.

The following facts are of importance in reference to precipitating-coefficients for caustic soda. If the lower sodium polysulphides are prepared either from lye of proper concentration, but not of sufficient temperature before adding the sulphur, or from diluted lye by boiling it with sulphur, reagents may result with precipitating-coefficients for caustic soda far below the normal values recorded in tables A and B. This can only be explained by assuming that a part of the caustic soda remains free and uncombined with sulphur, which is actually the case. For this and other reasons already stated, it is not judicious to attempt the preparation of a solution containing Na_2S in perceptible quantity in order to save sulphur. A solution in which Na_2S largely predominates will always have precipitating-coefficients below the normal values for caustic soda.

On the other hand, very concentrated solutions of the higher polysulphides may give precipitating-coefficients far above normal values. Since, however, these abnormal values, as already stated, are only obtained by partial precipitation of silver, copper, and lead from a hyposulphite solution, and do not exist in complete precipitation, and consequently not in practical mill-work, the subject is without economical value, and will not be discussed here.

Although great accuracy cannot be claimed for the tables below, regarding actual mill-work, they will be sufficiently near the truth

to base calculations of practical value upon them. They refer to freshly prepared sodium sulphide, and not to solutions oxidized by long contact with the atmosphere.

The following precipitating schemes may be considered of practical value.

A. Lead and calcium are absent, or only present in very small quantities.

Method 1.—Precipitation by Na_2S_2, or by CaS_5.

B. Lead and calcium are present in perceptible quantities, or lead alone is present.

Method 2.—Precipitation of lead by caustic lime, followed with precipitation of silver and copper by CaS_5.

Method 3.—Precipitation of lead and calcium by Solvay soda, followed with precipitation of silver and copper by Na_2S_2.

C. Calcium is present in considerable quantity, but not much lead.

Method 1, as given above, is here applicable; but in naming Na_2S_2 as precipitant under "C," it is distinctly understood that the reagent should be practically free from Na_2S, so that an oxidized solution contains only a minimum of Na_2CO_3. If it cannot be prepared with the amount of sulphur needed for the preparation of Na_2S_2 according to equation No. 1, more sulphur must be used.

In considering the economy of the different methods of precipitation, it would be misleading to rely only upon the figures given in the tables. There are other important elements that must enter into calculation, outside of the cost of chemicals, and the relative proportions of silver, copper, lead, and calcium in the solution. It is important, for instance, whether the lead-calcium carbonate can be profitably sold. If calcium predominates, the product may be exceedingly low in lead, and only salable at the expense of the silver it contains. An equally important factor is the grade of the sulphides in silver. The lower the grade, the heavier the freight and the smelting charges per ounce of silver, or the cost of refining if sulphides are treated at the mill. To this must be added increased expense in handling, namely, pressing, drying, sampling, and packing of precipitates. Evidently, general rules cannot be established regarding the most economical method of precipitation, but a calculation must be made for each individual case.

As repeatedly stated, tables for CaS_5 cannot be calculated on merely theoretical grounds. An attempt, however, is made in tables E and F, to base values for CaS_5 on actual mill-statistics.

A comparative test was made between Russell's sulphide and CaS_5, at the Cusihuiriachic mill, Mexico, as follows:

The mill, reducing 50 tons of ore per day, was run 38 days, using Russell's sulphide, and 21 days, using CaS_5 as precipitant. The quantities of chemicals consumed were as follows:

Using Russell's Sulphide:

Average value of ore: 35.1 ounces silver per ton.
Consumption of caustic soda: 4.4 pounds per ton of ore.
Consumption of sulphur: 2.9 pounds per ton of ore.

Using CaS_5:

Average value of ore: 39.0 ounces silver per ton.
Consumption of caustic lime: 24 pounds per ton of ore.
Consumption of sulphur: 10.3 pounds per ton of ore.

Outside of a difference in value, the ore was of exactly the same character in both cases. For better comparison, I reduce the above figures for CaS_5 to what they would have been in working ore of 35.1 ounces silver per ton, with the following result:

Corrected Values, Using CaS_5:

Consumption of caustic lime: 21.6 pounds per ton of ore.
Consumption of sulphur: 9.3 pounds per ton of ore.

To make these statistics more available for comparison, the composition of the sulphides should have been ascertained, besides the consumption in hyposulphite. Under the circumstances, I have to take the figures as they are. Considering what has been said about the preparation of CaS_5, and its deterioration by oxidation, the much higher consumption of chemicals involved in manufacturing CaS_5, compared with Russell's sulphide of equal efficiency, is by no means extravagant. To make comparison perfectly fair, we should add, however, for CaS_5 a gain of about 20 per cent. in hyposulphite, on a basis of the quantity of this salt contained in freshly prepared Russell's sulphide. For convenience in calculation, the addition of calcium hyposulphite is put down as its equivalent of the sodium salt.

In the financial table F, the price of caustic lime is taken at $\frac{1}{4}$ cent per pound, and other chemicals as stated in table D.

It is interesting to compare table F with table D. Even under

favorable assumptions for CaS_5, this reagent is, at stated prices for caustic soda, sulphur and sodium hyposulphite, very much dearer than Na_2S_2.

CaS_5 would be only cheaper where caustic lime can be obtained at much less than $\frac{1}{4}$ cent per pound, and where the cost of sulphur is abnormally low compared with that of caustic soda.

For this reason, in examining different methods of precipitation for a special case under normal conditions, we may just as well pay no attention to CaS_5 in Method 1, and discard Method 2 altogether.

Finally, if in lixiviation a considerable amount of sodium hyposulphite is consumed to keep up the strength of the stock-solution, it will be more profitable, in most cases, to use an oxidized sodium sulphide solution in place of one freshly prepared. Although the cost of precipitation is thereby apparently increased, the total cost of lixiviation will be diminished.

It has been claimed that in using CaS_5, the sulphides precipitate quicker and settle better than with sodium sulphide. This claim is not sustained by practical experience.

We find it stated in treatises on metallurgy that calcium hyposulphite is a more energetic solvent for gold than the sodium salt. This statement is without foundation in theory or practice. The solvent energy of calcium hyposulphite for silver-compounds is slightly inferior to that of the sodium salt, according to Russell's experiments.

The difference in deterioration of the two hyposulphites by atmospheric influences has already been mentioned.

TABLE A.—*Precipitating-Coefficients for c. p. Reagents.*

Calculations are based upon the following approximate chemical equivalents, frequently used:

$$H = 1;\ O = 16;\ C = 12;\ S = 32;\ Na = 23;$$
$$Ca = 40;\ Ag = 108;\ Pb = 207;\ Cu = 63.$$

Precipitating-Coefficients for NaHO, Consumed in the Manufacture of Na_2S_x.

100 NaHO precipitates 180.0 Ag as Ag_2S.
100 NaHO precipitates 172.5 Pb as PbS.
100 NaHO precipitates 105.0 Cu as Cu_2S.
Adding 103.3 $Na_2S_2O_3 + 5$ aq.

The following quantities of NaHO and S, consumed in the manufacture of Na_2S and Na_2S_2, are needed for the precipitation of metals:

For 100.	NaHO.	Sulphur for		Adding $Na_2S_2O_3 + 5$ aq.
		Na_2S.	Na_2S_2.	
Ag.	55.5	29.6	44.4	57.3
Pb.	58.0	30 9	46.4	59.9
Cu.	95.2	50.7	76.1	98.3

Precipitation of Lead and Calcium by Na_2CO_3.

100 Na_2CO_3 precipitates 195.3 Pb as $PbCO_3$.

100 Na_2CO_3 precipitates 37.7 Ca as $CaCO_3$.

For precipitation of

100 Pb are needed 51.2 Na_2CO_3.

100 Ca are needed 265.0 Na_2CO_3.

TABLE B.—*Precipitating-Coefficients for Commercial Reagents.*

Caustic soda with 90 per cent. NaHO, corresponding to the English rating of 70.6 per cent.; sulphur with 95 per cent. S; Solvay soda with 98 per cent. Na_2CO_3.

Precipitating-Coefficients for Caustic Soda, Consumed in the Manufacture of Na_2S_2.

100 Caustic soda precipitates 162.0 Ag as Ag_2S.

100 Caustic soda precipitates 155.2 Pb as PbS.

100 Caustic soda precipitates 94.5 Cu as Cu_2S.

Adding 93 $Na_2S_2O_3 + 5$ aq.

The following quantities of caustic soda and sulphur, consumed in the manufacture of Russell's sulphide and Na_2S_2, are needed in the precipitation of metals:

For 100.	NaHO.	Sulphur for		Adding $Na_2S_2O_3 + 5$ aq.
		Russell's Sulphide.	Na_2S_2.	
Ag.	61.6	41.0	46.7	57.3
Pb.	64.4	42.9	48.8	59.9
Cu.	105.7	70.4	80.0	98.3

Precipitation of Lead and Calcium by Solvay Soda.

100 Solvay soda precipitates 191.4 Pb as $PbCO_3$.

100 Solvay soda precipitates 36.9 Ca as $CaCO_3$.

For precipitation of
100 Pb are needed 52.2 Solvay soda.
100 Ca are needed 270.3 Solvay soda.

TABLE C.—*Weights of Precipitates and their Percentages in Metals.*

Weights of Sulphides.

From 100.	If precipitated by		
	Russell's Sulphide.	Na_2S_2.	CaS_5.
Ag.	124.3	129.6	174.1
Pb.	125.3	130.9	177.3
Cu.	141.6	150.8	227.0

Weights of Carbonates.

The precipitation by Solvay soda will produce from:

100 Pb, ... 129.0 $PbCO_3$.
100 Ca, ... 250.0 $CaCO_3$.

Percentages of Metals in Sulphides.

Percentage of	If precipitated by		
	Russell's Sulphide.	Na_2S_2.	CaS_5.
Ag in Silver Sulphide	80.4	77.1	57.4
Pb " Lead "	79.7	76.4	56.4
Cu " Copper "	70.6	66.3	44.0

Percentages of Metals in Carbonates.

Lead carbonate, 77.5 Pb.
Calcium carbonate, 40.0 Ca.

TABLE D.—*Financial Results.*

Showing the cost of precipitation by Russell's sulphide, Na_2S_2, and by Solvay soda, assuming the following prices for commercial chemicals put down at the mill: caustic soda, 5.5 cents; sulphur, 3.5 cents; Solvay soda, 4.0 cents; hyposulphite, 4.0 cents per pound.

Cost of Precipitating Metals as Sulphides.

For 100 Pounds.	By Russell's Sulphide.	By Na_2S_2.	Gain in $Na_2S_2O_3 + 5$ aq.	Net Cost using		Difference in cost between Russell's Sulphide and Na_2S_2.
				Russell's Sulphide.	Na_2S_2.	
Ag.	$4.82	$5.02	$2.29	$2.53	$2.73	$0.20
Pb.	5.04	5.25	2.40	2.64	2.85	0.21
Cu.	8.28	8.61	3.93	4.35	4.68	0.33

Cost of Precipitating Lead and Calcium as Carbonates.

For 100 pounds Pb, $2.09.
For 100 pounds Ca, 10.81.

TABLE E.—Showing the quantities of caustic lime and sulphur consumed in the manufacture of CaS_5, needed for the precipitation of metals.

For 100 Pounds.	Caustic Lime.	Sulphur.	Adding $Na_2S_2O_3 + 5$ aq.
Ag.	302.4	130.2	68.8
Pb.	316.1	136.1	71.9
Cu.	519.0	223.4	118.0

This table is based on comparative statistics obtained in the Cusihuiriachic mill, Mexico.

TABLE F.—Showing the cost of precipitation by CaS_5, assuming price of caustic lime at $\frac{3}{4}$ cent per pound, and that of other chemicals the same as given in table D.

Cost of Precipitating 100 Pounds.	Caustic Lime.	Sulphur.	Total.	Gain in $Na_2S_2O_3 + 5$ aq.	Net Cost.
Ag.	$2.26	$4.56	$6.82	$2.75	$4.07
Pb.	2.37	4.76	7.13	2.88	4.25
Cu.	3.89	7.82	11.71	4.72	6.99

This table is based upon the figures given in table E.

§ 7. *Application of the Tables to Special Cases.*

In this section the tables will be applied to examine the cost of precipitation in special cases. It is assumed that the ore contains lead and a small amount of calcium. Although its contents in copper are insignificant, the copper from extra-solution, necessary for treatment, comes to precipitation. We will neglect the comparatively small amount of silver precipitated with the lead-calcium carbonates, assuming that such silver is paid for at the same rates as silver in sulphides. Prices of chemicals are taken as stated in Table D.

We further assume the following terms, according to a case as it really exists, for the disposal of the sulphides to smelting-works and the cost of handling the precipitates:

The Marsac mill, according to W. A. Wilson, sells its sulphides to the Omaha & Grant S. & R. Co., the latter paying for 97 per cent. of the silver, New York quotations, charging $30 per ton for treatment, but paying freight from Park City to Omaha. No allowance is made for copper or lead. [Gold is paid for at the rate of $20 per ounce.]

Carbonate precipitates are taken at the same rates, as also the low-grade precipitates from the first wash-water. It will be seen at once that the disposal of carbonates at sulphide rates cannot be favorable to the separate precipitation of lead in the presence of calcium. The Marsac mill used this method formerly for a short time only, then abandoned it, but has adopted it again recently. A better market for carbonates will, no doubt, be found hereafter.

The cost of handling precipitates is assumed at $10 per ton.

Example.

The solution obtained in lixiviation contains per ton of ore:

3 pounds (43¾ ounces) silver.

1 pound copper.

3 pounds lead.

½ pound calcium.

Four pounds sodium hyposulphite have to be regenerated from tetrathionate in precipitation. This expense, however, we need not consider, since it is the same for Methods 1 and 3.

I hardly need remind the reader that in precipitation Na_2S_2 free from Na_2CO_3 is used, and not an oxidized solution of Russell's sulphide, which would also precipitate calcium in Method 1, changing the weight and grade of the sulphides.

Weights of Precipitates.	Precipitation by	
	Method 1.	Method 3.
Sulphides,	9.32 pounds.	5.40 pounds.
Carbonates,	.....	5.12 "
Sulphides contain		
Ag, per ton, oz.,	9388.5	16,201.7
Cu, per cent.,	10.73	18.52
Pb, " "	32.19	.....
Carbonates contain Pb, per cent. 58.6.		
Cost of	Cents.	Cents.
Precipitation,	21.42	25.45
Handling,	4.66	5.26
Reduction,	13.98	15.78
Total,	40.06	46.49

It will be seen at once that separate precipitation of lead and calcium cannot be profitable if carbonates are sold at sulphide rates. Also, that Method 3 becomes more costly, the more calcium is present in proportion to the lead.

We will now assume 90 per cent. of the lead in carbonates to be paid for at the rate of 2 cents per pound, and freight and smelting-charges to be $15 per ton; also that silver is paid for at sulphide-rates. This would give the following result:

	Cents.
Precipitation,	25.45
Handling,	5.26
Reduction, sulphides,	8.10
Reduction, carbonates,	3.84
Total,	42.65
Less amount realized from sale of lead,	5.40
Total,	37.25

Should the buyer, however, deduct 5 per cent. of the silver in

place of 3 per cent., nearly all the profit from the sale of lead may be consumed, or even a negative balance left, depending on the richness of the carbonates in silver.

It is hardly necessary to continue this discussion. The separate precipitation of lead and calcium by Solvay soda is evidently not so profitable as its advocates claim. If lead alone is present, or the amount of calcium is very slight in proportion to lead, Method 3 can be used to advantage.

It is claimed that the removal of calcium from the lixiviation-solution has a beneficial effect on the extraction of the silver. Whether this is true, I do not know.

If sulphides are refined at the mill by a humid process with sulphuric acid, the absence of lead and calcium is essential to success; and in this case Method 3 is of great value, as I shall demonstrate in another paper.

My friends who operate lixiviation-works will, most likely, object to many conclusions I have drawn in this treatise, saying that the process as carried out in practice is not like that operated here on paper. If they will bring out an array of facts, proving that I am wrong, I shall consider myself the gainer, earning the thanks of the profession for bringing information to light which otherwise would have remained hidden.

APPENDIX.

The Loss of Hyposulphite in Lixiviation.

In connection with precipitation, some remarks about the loss of hyposulphite in lixiviation will be pertinent.

As has been shown, large amounts of hyposulphite are added to the stock-solution in precipitation, even with freshly prepared calcium or sodium sulphides. From this it might be expected that the stock-solution would always increase in strength and concentration. This is only the case if, in ordinary lixiviation, CaS_5 is used as precipitant, and comparatively large amounts of copper and lead are thrown down together with the silver.

The losses in hyposulphite are partly mechanical, and partly caused by chemical decomposition. The mechanical losses take place when the first wash-water is replaced by stock-solution, and again when the latter is replaced by the second wash-water. In these operations, no matter how carefully they are conducted, water and stock-solution are more or less mixed, especially where the

filters are not in good condition and are partly choked. Thus, in replacing the first wash-water, a certain quantity of hyposulphite solution results, too weak to be mixed with the normal solution; for this reason it is transferred to a separate precipitating-tank, precipitated there by itself, and the decanted clear solution run to waste. As soon as the solution shows a certain strength in hyposulphite, which is ascertained by the iodine-test, it is turned into a regular precipitating-tank. The same is the case in replacing the solution by the second wash-water. The higher the percentage of hyposulphite in the stock-solution, the greater this mechanical loss.

The chemical losses are caused as follows:

1. By oxidation in contact with the atmosphere, hyposulphite being converted into sulphate. This loss depends on the quantity of solution used per ton of ore and kept in rotation, its temperature, and upon atmospheric conditions, most likely the contents of the atmosphere in ozone.

2. By converting hyposulphite into tetrathionate in preparing extra-solution, and further formation of tetrathionate by atmospheric oxidation of extra-solution. This loss is, however, made largely good again by close precipitation, and may be almost completely covered by over-precipitation.

3. By decomposition of extra-solution with formation of Cu_2S and sulphates; this loss being final and irretrievable. It is greatest where extra-solution has to perform much work—*i.e.*, where standard extra-solution must be used warm and is circulated It is, no doubt, greater if a Koerting ejector is used for circulation than with a geyser-pump. This subject is discussed in my paper, "The Construction of Details for a Modern Lixiviation Plant." (Page 3 of this volume.)

THE REFINING OF SULPHIDES OBTAINED IN THE LIXIVIATION PROCESS WITH HYPOSULPHITE SOLUTIONS.

BY C. A. STETEFELDT, SAN FRANCISCO, CAL.

(Cleveland Meeting, June, 1891.)

STEAM-DRIED sulphides, obtained in the lixiviation-process, are now almost exclusively sold to smelters, the old methods of melting (after roasting) in crucibles, or cupelling with lead at the mill, having

become obsolete, principally because they involve accumulation of a large quantity of silver in by-products, that must be sold to smelters under any circumstances.

It would be desirable to refine sulphides at the mill by a humid process, especially when they contain a large percentage of copper, which would be converted into sulphate, the latter being used again in the mill for preparing Russell's extra-solution. If refining is done by a humid process, it is hardly necessary to say the sulphides should be nearly free from lead and calcium,* and these metals, if present in the lixiviation-solution in perceptible quantities, should be precipitated separately by Solvay soda.

The sulphides should also be free from soluble salts contained in the hyposulphite stock-solution, since these salts cause serious difficulties in roasting and contaminate the copper sulphate obtained as a by-product. Sulphide cakes from the filter-press may hold 50 per cent. and more of moisture. If the sulphides are not washed before going to the filter-press, or in the latter itself†—which practice mostly prevails now—they may contain, after steam-drying, a considerable quantity of sodium hyposulphite, sulphate and chloride, besides some other salts. While the percentage of sodium hyposulphite in the stock-solution is kept constant, that of other salts increases to a certain limit, which may be very high. In one sample of Marsac sulphides, 11.3 per cent. of these salts was found, and in another, 13.2 per cent. Under all circumstances it is bad practice not to wash sulphides. Steam-drying becomes more difficult and expensive, the salts retaining water of crystallization, and if sulphides are sold to smelters,‡ freight and working-charges have to be paid on worthless material. Besides, the sulphides may take up moisture during storage and shipment, thus causing disputes about discrepancies in weight and assay-value.

The whole problem of refining sulphides at the mill presents financial as well as technical difficulties, *i.e.*, commercial success depends on the location of the works.

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* Concerning calcium in sulphides, see my paper on "The Precipitation of Metals from Hyposulphite Solutions." (Page 15 of this volume.)

† Precipitates are washed easily, and with a minimum of water, in a filter-press especially constructed for that purpose. In washing sulphides a very small quantity of Na_2S_2 should be added to the wash-water to prevent sulphatization of Cu_2S.

‡ The freight and smelting-charges on Marsac sulphides are $30 per ton; hence, in not removing the soluble salts, from $3.39 to $3.96 are paid for shipping and reducing worthless material that might have been just as well left at the mill.

In one of my papers (*Trans.*, xiii., p. 105) I proposed to dissolve raw sulphides in concentrated, nitrated sulphuric acid, with regeneration of the escaping nitric oxide to nitrous and nitric acid, absorbing the latter by concentrated sulphuric acid. I am now of the opinion that this process is without merit. The principal advantage obtained for it was that roasting the sulphides, with loss of silver, might thus be avoided. But if steam-dried sulphides are carefully roasted in a muffle-furnace (the muffle being heated only from above, not from below), and the escaping gases are drawn through a Roessler convertor, the loss of silver by volatilization can be entirely precluded.

In my book, *The Lixiviation of Silver Ores with Hyposulphite Solutions*, I described a method essentially the same as that used by A. D. Hodges (*Trans.*, xiv., p. 731) for refining sulphurized, base Comstock bullion, with this difference, that in my plan the sulphuric acid for dissolving copper oxide was to be obtained as a by-product by drawing the sulphurous acid from the muffle-furnace through a Roessler convertor. This method, which is only outlined in my book, I shall now consider more in detail. The principal difficulty of the process centers in the roasting. If sulphides are roasted as they come from the drying-chamber, after grinding the lumps in a sampling-mill, the resulting product is unsuited to treatment with dilute sulphuric acid. No matter how carefully the roasting is done, a granular mass is obtained, consisting largely of metallic alloys, with comparatively little copper oxide. It is not difficult to discover the reasons for this behavior.

1. In burning the free sulphur, the charge commences at once to form little balls.

2. The sulphides are not in the form of a copper-silver matte, but consist of individual minute particles of the sulphides of silver and copper.

3. Silver predominates too much over copper.

4. Silver sulphide oxidizes too rapidly, and the copper oxide formed immediately reacts upon remaining sulphides, converting them into metals. Most likely, even at very low temperature, actual fusion of the very fine particles takes place, favoring the above reaction. By continued oxidation, a portion of the copper in the alloy could, no doubt, be converted into oxide, but the preponderance of silver would prevent oxidation to any depth. The same may be said concerning the action of the Roessler converter. In the latter, metallic copper is readily dissolved by dilute sulphuric acid; but it

is not probable that this would be the case with an alloy containing so much silver.

As already mentioned, lixiviation-works now ship their product to smelters unroasted. Although, by roasting sulphides, their weight is materially lessened, and the product is obtained in a better and less bulky form for packing and shipping, the objection to roasted sulphides is based upon the difficulty, almost impossibility, of their correct sampling, making a fair settlement between buyer and seller a source of much annoyance.

All difficulties in roasting can, however, be overcome if we convert the sulphides into a matte, at the same time incorporating more copper, in the shape of cement-copper, in order to obtain a product which permits a good Ziervogel roasting. There is more than enough free sulphur in the sulphides to sulphurize all the copper necessary for this purpose.

Hodges sulphurized base Comstock bullion easily and at a trifling expense in a cast-iron kettle. I see no reason why the matting and copperizing of sulphides could not be done just as well in the manner described by Hodges. He succeeded in roasting a copper-silver matte, which contained on an average equal parts of copper and silver, converting as much as 75 per cent. of the silver (on an average, 60 per cent.) into silver sulphate. From the roasted product the copper was almost completely dissolved as oxide by chamber-acid.* Most probably a copper-silver matte may be successfully roasted, even if silver predominates slightly over copper. But assuming that the relation of silver to copper must be one to one, the amount of copper needed would not be very large where ores are treated by the Russell process, and a comparatively large amount of copper comes to precipitation. All this copper would be returned as sulphate.

The operations of the process would be as follows:

1. Matting and copperizing the sulphides in a cast-iron kettle. Sulphurous acid, produced by burning of an excess of sulphur, should be conducted to the muffle-furnace and thence to the Roessler converter, or to the latter at once. The matte may be removed from the kettle while in a pasty condition, *i.e.*, before solidifying.

2. Pulverizing of the matte through a No. 40 screen.†

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* The bars produced from sulphurized so-called "white bullion" were on an average .967 fine in silver and gold.

† One cubic foot of matte, pulverized through a No. 40 screen, will weigh about 175 pounds.

3. Roasting in a muffle-furnace, with a Roessler converter attached.

The roasting should be conducted as was done by Hodges, *i.e.*, for the formation of a maximum of silver sulphate. Should the sulphuric acid produced in the converter be insufficient for dissolving all the copper oxide, pyritic ores, containing silver, could be roasted occasionally, at slight expense, to make up the deficiency; or sulphur could be burnt should this prove more economical.

4. Boiling the roasted sulphides (after separating sintered lumps by screening) in a lead-lined tank with the sulphuric acid obtained in the Roessler converter.

In this operation either of two methods may be selected. One, as described by Hodges, consists in the separate precipitation of the silver sulphate from the solution by copper, if it is desirable to raise the fineness of gold in the insoluble silver residue; the other, in effecting the decomposition of silver sulphate at once by placing copper plates in the dissolving-tank. The latter method is the simpler, requiring less apparatus and manipulation, and deserves preference when gold is absent, or present in sufficient quantity to stamp the bars.*

5. Crystallizing the copper solution, if necessary, after concentration.

Since all or most of the copper sulphate is used again in the mill, and not for sale, special attention need not be paid to the production of a fine-looking or very pure article. In fact, small crystals are more desirable for the preparation of extra-solution than large ones. Mother liquor from the crystallizing-vats would be returned to the Roessler converter.

The concentration of solutions I propose to effect by running them through a lead-tower, with inclined shelves, against a current of hot air. The latter would be drawn from the chimney of the muffle-furnace by a Koerting steam-jet ventilator.

6. Washing of the silver residues (and cement-silver); pressing into cakes; drying and melting into bars.

7. Precipitating cement-copper by scrap-iron from wash-waters,

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* In connection with this point, I made the following experiment: An alloy of pure silver and copper, .500 fine in silver, was boiled with silver sulphate. Cement-silver was rapidly precipitated, although much more slowly than by the use of pure copper. I then took Ontario bullion, .440 fine, which is principally an alloy of silver and copper, and immersed a sheet of it in a silver sulphate solution. The reduction to cement silver was very slow and ceased entirely after a short time. Probably the presence of a small amount of lead interfered with the reaction.

weak solutions and from mother liquors in which too much iron has accumulated.

In some cases a surplus of copper sulphate over that needed for extra-solution in the mill may result. Where this could not be sold at a profit, it should be precipitated by scrap-iron, thus keeping in rotation the cement-copper for copperizing the matte.

It will be seen that the process in this form presents no essential technical difficulties, all its operations being well understood by metallurgists; nor does it require large, expensive or complicated apparatus. The refinery should be built on a scale to remain in operation and keep the men employed continuously during the day-time only.

Whether it will be more profitable to refine sulphides at the mill or sell them to smelters, must be calculated for each individual case and depends entirely on local circumstances.

I have no doubt that refining sulphides at the mill would extract a higher percentage of the precious metals, especially of the gold, than is shown by the commercial assay upon which settlement is made between the buyer and seller. Smelting-works must have a margin in their favor, or they could not conduct their business with a sufficient profit at the rates they charge for treatment and the percentage they deduct from the commercial assay-value. Even the Freiberg works, belonging to the government, have their *Huettenremedien* and extract so-called "plus" silver.

Having in view the erection of a refinery for sulphides at the Marsac mill, Utah, I made the following laboratory experiments.

Experiments with Sulphides Containing Lead and Calcium.

(*Sulphides Unwashed.*)

The lot of Marsac sulphides, a sample of which was at my disposal, was said to have been produced after the precipitation by Solvay soda of lead and calcium from the solution; but either the precipitation of these metals was not complete, or the sulphides had become mixed with those obtained prior to the introduction of separate precipitation for lead and calcium. The sample contained 34.7 per cent. silver (9.0 ounces gold per ton), 19 per cent. copper and 2 per cent. lead; calcium and other constituents not determined. A few grammes of these sulphides were digested with dilute hydrochloric acid. Carbonic acid and sulphuretted hydrogen were evolved,

showing the presence of calcium carbonate* (and lead sulphide). The filtered solution was boiled with potassium chlorate, after which lead and a small amount of iron were precipitated by ammonia. From the filtrate a very perceptible precipitate of calcium oxalate was obtained. Most of the calcium however, remained, with the residue as sulphate.†

In roasting these sulphides in a clay dish in the muffle of a cupelling-furnace the free sulphur was burnt quickly, and then a rapid roasting at a very low temperature took place. In elevating the temperature as required for a dead roast, the sodium sulphate (during roasting, sodium hyposulphite and chloride are also converted into sulphate) commenced to fuse, and the final result was a granular mass of brittle, metallic globules, on which continued roasting did not seem to have much effect. On treating them with sulphuric acid of 15° B., not a trace of silver sulphate was found in solution, and after continued boiling only a small quantity of copper was extracted.

Now, 50 grammes of sulphides were mixed with 8 grammes of cement-copper, to raise the percentage of copper to that of silver, and heated in a small cast-iron crucible. Some sulphur fumes escaped, and then quick and complete fusion took place at a low red heat. The very liquid matte was poured out. Not a particle of metal had separated, and the inside of the crucible was not visibly affected. The matte, pulverized and sifted through a No. 40 screen, was roasted at a temperature just sufficient to ignite it. It behaved very well and did not bake in the least just then. But when the temperature was raised, after the blue flame of sulphur had disappeared, the sodium sulphate commenced to fuse, and the roasting-dish was filled with a mass like mush. Under still further increase of heat, the charge commenced to foam, through decomposition of sulphates and reaction between oxides and sulphides, and shortly after it hardened into solid, porous crusts. These were pulverized and boiled with dilute sulphuric acid. Rapid solution of the copper

* Concerning the presence of calcium carbonate in sulphides, see my paper, "The Precipitation of Metals from Hyposulphite Solutions," already cited.

† It is not astonishing that so little calcium carbonate should be found in and dissolved from the unwashed sulphides. In digesting the sulphides with dilute hydrochloric acid, the sodium sulphate present precipitates most of the dissolved calcium as gypsum. In subjecting the sulphides to steam-drying, there is also opportunity for converting calcium carbonate into sulphate. Gypsum as such cannot be precipitated by Na_2S_2 from a sodium hyposulphite solution (as constituted after use in the mill) containing calcium.

oxide took place, and from this solution copper sulphate and a great deal of sodium sulphate crystallized. The metallic residue was fused with borax, yielding silver .790 fine. It can be seen, as observed at the beginning of this paper, that roasting and treatment as above of unwashed sulphides is not practicable. Under such circumstances the roasting is not only very difficult and incomplete, but the resulting silver is too low in fineness and the copper solution is contaminated with sodium sulphate.

The sulphides were now leached with water, and the purified product was subjected to the same treatment as formerly.

Experiments with Sulphides Containing Lead and Calcium.

(Sulphides Washed.)

The direct roasting of washed sulphides proceeded better than that above described; but when the temperature was raised the charge became rather sticky, foamed and then hardened. It contained considerable copper oxide; and, after boiling with sulphuric acid, the insoluble residue, melted with borax, yielded a button about half metal and half matte. The matte contained 35 per cent. of silver and the metal was .897 fine.

After matting and copperizing a sample, as described before, I found that roasting could be carried on without the slightest difficulty, the charge behaving well from beginning to end. I made no attempt at Ziervogel roasting, which is rather difficult anyhow with small samples, and with material containing so much lead and calcium. The temperature was quickly raised after most of the sulphur had been oxidized. At the end the charge was kept at a good red heat. After boiling with acid, the residue yielded a bar .938 fine in silver. With slower and more careful roasting, finer silver would, no doubt, have been produced, although the presence of lead and calcium sulphate must also have exerted an unfavorable influence.

Experiments with Sulphides Entirely Free from Lead and Calcium.

Having, at the time, no sulphides free from lead and calcium at my disposal, I produced them by converting 10 grammes of silver into chloride, dissolving the latter in sodium hyposulphite solution and adding 40 grammes of $CuSO_4 + 5\,aq = 10$ grammes Cu. The solution was precipitated with sodium sulphide and the sulphides were washed, dried and matted.

In roasting the pulverized matte great care was taken to regulate the temperature so as to form a maximum of silver sulphate, especially by not allowing the charge to get too hot at the end of the roasting. This was accomplished remarkably well. In dissolving the roasted matte nearly all the silver went into solution and was precipitated as cement-silver. The final result was a bar .988 fine in silver.

Experiments with Sulphides Containing only Small Quantities of Lead and Calcium.

(Sulphides Washed.)

Finally, I obtained from the Marsac mill samples of a lot of sulphides containing only a trace of lead and much less calcium than the first lot.

The contents in silver were 34.6 per cent. (8.6 ounces gold per ton) and in copper, 20.0 per cent. Soluble salts, 13.2 per cent. After leaching out soluble salts, a part of these sulphides was matted directly and another part with incorporation of cement-copper, as described for former experiments. Both samples were now roasted with the intention of forming as much Ag_2SO_4 as possible. Only a small portion of the silver, however, was converted into sulphate. This was evidently due to the presence of calcium in the matte.

The sample to which copper had not been added was rather difficult to roast, requiring constant stirring and showing a tendency to sinter, while the copperized matte behaved very well, as described in previous experiments. The addition of copper is of decided advantage in facilitating the roasting, and, in consequence, raising the fineness of the silver after solution of the copper. The matted and copperized sulphides yielded silver .990 fine, while the sulphides matted without addition of copper yielded silver only .938 fine.

The roasting in all these experiments was done in the muffle of a cupelling-furnace which held clay-dishes of only 3½ inches diameter. In consequence, the roasting of such small masses proceeded too rapidly, which was detrimental to the formation of silver sulphate, and favored the separation of metallic silver alloyed with more or less copper. There is no doubt that finer silver will result after careful roasting on a large scale. Sintered lumps, containing unroasted matte, should, of course, be separated by screening, and, after pulverizing, should be re-roasted with the next charge. The presence of small quantities of lead and calcium is evidently not injurious, although it interferes with a good Ziervogel roasting.

This paper would have possessed more value and interest if the experiments had been followed by analysis of the materials and all the products. To do this I had neither the facilities nor the time; besides, such work is done more profitably at the mill, for reasons that require no explanation. I hope the muscular lixiviators will wake up to the fact that their work cannot prosper without analytical help. The present condition of affairs is disgraceful, as has been more fully shown in my paper on "Precipitation of Metals." (Page 15 of this volume.)

I refrain from giving estimates of the profit and loss in refining sulphides by the process described above, hoping to do this at some future time, after its introduction in practice.

GEOLOGICAL NOTES ON THE MANGANESE ORE-DEPOSIT OF CRIMORA, VIRGINIA.

BY CHARLES E. HALL, CHICAGO, ILL.

(Cleveland Meeting, June, 1891.)

THE Crimora mine, operated by the American Manganese Company, Limited, is located 2½ miles east of Crimora station, on the Shenandoah Valley railroad, in Augusta county, Virginia. It is situated in a basin at the western base of the range of mountains which flanks the Shenandoah Valley on the east. This range is composed of the Potsdam sandstone (No. I. of the Pennsylvania survey), and extends in a northeasterly-southwesterly direction the entire length of the valley. The general dip of the measures is westward. The massive sandstone of the mountain is geologically overlaid with shales or slates, which belong to the Potsdam, or lie between it and the Lower Silurian limestone. Throughout this region these shales are decomposed, and form more or less extensive clay deposits, which are found along the base of the range and often underlie a considerable area to the westward of the abrupt slope of the range. Sand and fragments of quartzite occur in considerable quantities on the surface, and often conceal the true character of the underlying formation.

The area containing this ore-deposit extends northeast and south-

west, and is flanked on the east by the main range and on the west by a sharp anticlinal flexure of the sandstone, which, to the north of the ore-deposit, forms a prominent spur of the range. Figs. 1 and 2 will illustrate the statements given in this paper.

Fig. 1 is a ground-plan of the immediate vicinity of the mine. The line AA is the general line of axis of the monoclinal of the Potsdam sandstone, with westward dip. BB indicates the synclinal axis, which sinks rapidly to the southward. CC is the line of the anticlinal of the Potsdam. Near the centre of the figure is indicated

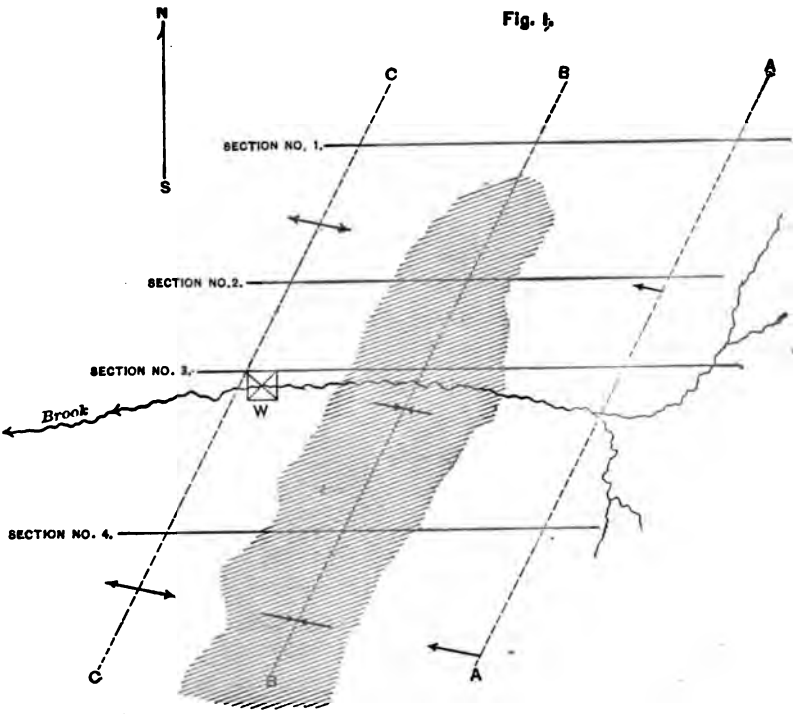


Fig. 1.

Plan of the Vicinity of the Crimora Manganese Ore-deposit.

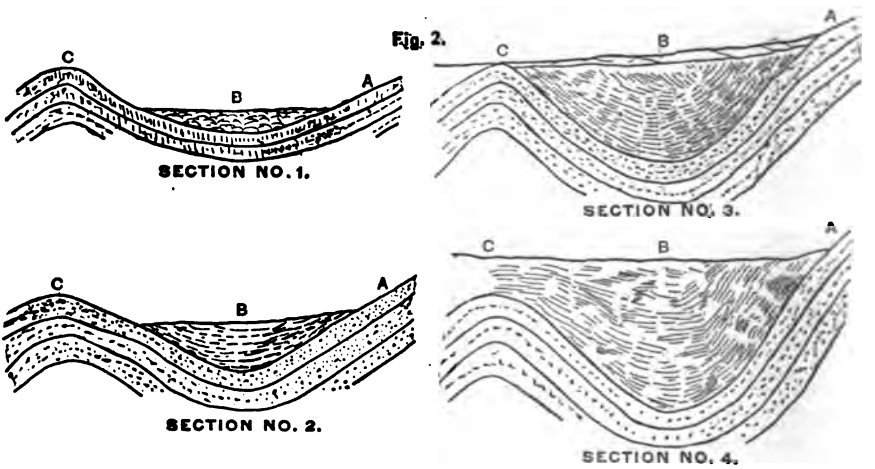
a brook flowing from the monoclinal ridges of the Potsdam on the east and crossing the synclinal basin and also the anticlinal fold CC to the westward. W indicates the location of the ore-washer. The sections, Nos. 1, 2, 3 and 4, Fig. 2, are taken on lines indicated by the respective numbers in Fig. 1.

It will be seen from Fig. 1, that east of the mine there is a large area drained through the brook which crosses the ore-basin. This drainage-area is principally within the Potsdam sandstone-belt. The

brook leaves the synclinal (ore-bearing) basin at the point where the Potsdam anticlinal fold, indicated in the sections by C, sinks below the surface. It will be seen from the sections, Fig. 2, that the axis C, as well as B, rapidly sinks to the northward.

The clay resulting from the decomposition of the shales has been preserved within this sharp synclinal. In section 1 scarcely any clay appears, while between sections 3 and 4 more than 300 feet of clay has been penetrated in boring for ore. The shaded portion of Fig. 1 indicates the ore-area, which extends irregularly northeast and southwest with the axis of the basin. The distance across the basin is one-fourth of a mile. The drainage-area of the brook, west of the basin, is fully 10 square miles.

The fold C, section 3, forms a complete dam, back of which the



Sections through Crimora Ore-basin (see Fig. 1).

seepage-water is held until it can slowly work its way through the sandstone of C, or southward and past the end of the sinking anticlinal axis.

Black oxide of manganese occurs in the Potsdam sandstones of this region in greater or less amount. It is often noticeable as a black stain on the rocks, or as minute specks or particles throughout the mass. No workable bodies of this ore have been found within the Potsdam sandstone, so far as I am aware.

There are no limestones west of, or in proximity to, the manganese ore-deposits of Crimora which could give origin to the ore; and I am therefore forced to the conclusion that the ore in this case origi-

nates from the Potsdam sandstone, where it exists disseminated throughout the rocks. The water draining from the mountain area east of the clay-basin carries the mineral with it in minute quantities, and redeposits it in the clay which fills the basin B. This argument is favored by the fact that the largest and best deposits of ore have been discovered directly beneath the course of the brook and immediately upon its entrance to the clay-area.

The ore is found in irregular bodies in various parts of the clay mass, and I am unable now to say what are the most favorable conditions for its deposition. Clay seems to be essential to its formation or re-deposition. It is seldom found in sandy ground, and when it is thus found, one is led to the belief that clay originally formed part of the mass, and has been subsequently washed out. A piece of sandstone imbedded in clay will sometimes become so much impregnated with the black oxide of manganese as to be in fact a siliceous manganese-ore. To my mind it appears that the water, laden with its particles of manganese minerals, becomes retarded upon reaching the clay mass, and a separation takes place more or less rapidly. The rate at which this separation or re-deposit takes place is difficult to estimate, but it is undoubtedly going on at the present time. Where the water is kept artificially drained from the mines, we cannot look for the natural conditions to exist; but if the natural conditions are permitted to return, the formation of the ore will continue.

There appears to be no ratio between the percentages of phosphorus and iron in manganese-ore. In cases where the iron percentage is low, the phosphorus may be high, and *vice-versa*. I will therefore not attempt to enter into any genetical explanation. Nor am I prepared to generalize on the subject of manganiferous ore-deposits. The facts which I have here endeavored to explain apply to the deposit in question, and may not be applicable to any other. Similar conditions, however, may exist in other sections; and it would be advisable for those interested in prospecting for manganese-ore to bear in mind the geological features which I have found here.

THE ALLUVIAL TIN-DEPOSITS OF SIAK, SUMATRA.

BY CHARLES M. ROLKER, E.M., NEW YORK CITY.

(Cleveland Meeting, June, 1891.)

THE main tin-producing regions of the world are known to be England, Australia and the Dutch East Indian possessions, chiefly Banca and Billiton. During recent years, the tin of the Malay Peninsula, exported from Penang and Singapore, has been coming prominently to the front. France, Austria, Hungary, Germany, Spain, South America, Russia and China still present insignificant figures.

The conspicuous part which the Dutch possessions in the Indian Archipelago have taken in supplying the world with tin, is well known. Tin was discovered in Banca as early as 1710, and has been produced there ever since; but the supply has been most important since Banca came fully into the possession of Holland, in 1821.

In Billiton and the adjoining island of Singkep, the existence of tin was known prior to 1822.* The Lingganese knew of its existence in the valley of Embalong, from creeks and streams in the neighborhood of Tima and Simper hills, on Billiton and also on Singkep island, but the prevailing superstition that tin-mining brought misfortune, influenced them to let this industry lie idle. The attempts of the Dutch government to test this question were lukewarm, perhaps by reason of the fear of reducing, through new discoveries, the price of Banca tin, and the island was abandoned in 1826, leaving only a military post.

In 1850, Dr. Crookewit, a chemist and naturalist, was sent out to verify the rumors that tin existed in Billiton. His report was unfavorable; but in 1851 Mr. C. De Groot, a mining engineer, found tin readily, and the active work on Billiton dates from that time. Less than 500 tons were produced during the first ten years, and 3000 tons in the next decade. Billiton tin-mining is done by

* Dr. Th. Posewitz, *Petermann's Geographische Mittheilungen*, 1857, p. 108.

private parties, while 70 per cent. of the mines of Banca are controlled by the Dutch government; formerly the government controlled all the Banca mines.

The Banca metal is shipped to the island of Java, and exported from the port of Batavia to Holland. It figures in the markets as tin exported from Batavia.

In the Malay Peninsula the Chinese are known to have worked tin-diggings as early as 1793. Since then the development has been gradual, until a comparatively recent date. In 1866 the exports of tin metal (white metal) amounted to 5692 tons; in 1874 the total was 13,566 tons; in 1883 it was 17,195, and rose to 28,492 tons in 1889, an increase of 65 per cent. This shows the extensive development of tin-mining in these parts of the world. The energy displayed is great, in view of the natural conditions of the country, and the disadvantages under which work is prosecuted.

New tin-fields will unquestionably be found in the Peninsula, as well as within that belt of countries or islands which lies between Australia on the south and the Malay Peninsula on the north—known as the Sunda, and their adjacent islands—but so far no considerable production can be traced from any of the latter districts outside of Banca and Billiton. Prospecting and developing the islands referred to will be slow and difficult, because of their dense tropical vegetation, unfavorable climate for labor, and a lack of facilities for such enterprises.

The tin-deposits of Banca and Billiton—the types of the Indian Archipelago—are mainly alluvial, although some lodes, supposed to be of recent origin, have been worked.

According to Davies the bed-rock of the country is granite, metamorphosed slates, quartzites and sandstones. A cross-section through the Banca deposits would show, following from bed-rock upwards, an average of 3 feet of tin-ore overlaid with coarse sand, followed by clay (red, white and black), then coarse sand, with pockets of clay and layers of fine sand carrying a little tin-ore; then humus. The average overburden is 25 to 35 feet thick. Similar conditions are said to exist on the Singkep, Carimon and other islands, where, so far, only desultory prospecting has been done, without financial success, or where companies have only lately been organized for work. I learn that tin has been found as far east as the island of Floris in the Floris Sea.

The tin of Banca and Billiton has been traced to its original sources in the granite and gneiss; it is also found cutting through the

supposed overlying quartz-schist, clay-slates and clayey sandstones, but, so far as I can learn, very little lode-mining has been profitably done.

In the Malay Peninsula tin-lodes have been found, in addition to alluvial tin, and preparations are in progress to test their value.

The system of working the tin-alluvions throughout these districts, is what is known as the "Chinese system," of which I shall say more later on.

The attention of capital was lately directed to a tin-field on the Island of Sumatra and within the district of Siak. It lies inside of a large tract of land, estimated by the promoters to contain 400,000 to 500,000 acres, conceded by the Sultan of Siak, Iri Indrapoera, in 1887, to Dutch parties, "for the extraction of tin and other minerals which appear in that part of the dominions." I quote the official limits of this concession to define the geographical position of what is considered the tin-belt of East Sumatra. The ground conceded by the Sultan is bounded "on the north by the part of an imaginary line running from Loeboehbandahara, on the Rohkan river, as far as Tandjong Maris to the Tapong Kanau river (in Siak) and further by the Tapong Kanau river; on the east by an imaginary straight line drawn from the southern boundary of Siak, straight from south to north, and running through Batoe Gadja, as far as the Tapong Kanau; on the south by the independent states to Kampang Kanau and the Podang Highlands; and on the west by the Rohkan States." These are the official boundaries of the tin-fields, based on the knowledge of the government. Since all inhabitants working the tin-deposits are obliged to pay a tax to the Sultan on the tin produced, it is to be assumed that the government had a guide in thus delineating the confines of the prospective tin-fields, in addition to its knowledge of the work of an exploring party in 1881.

Sumatra is divided longitudinally, by volcanic ranges, into the east and west coast-country. The west coast is a narrow strip of land, 20 to 30 miles wide, underlaid, according to Dutch geologists, with granite; in places it is covered with alluvium and coral formation, sandstones, slates, and volcanic rocks. Marble and coal are found near Podang, and the precious metals are mined in other parts. Several railroads extend from the coast to the foot of the mountains above mentioned. East Sumatra is a generally low and slightly undulating country, of the geology of which little is known. Marshy districts abound, in consequence of the extensive flood-

plains of the large rivers. The rivers rise in the divisional range and flow in a tortuous course to tide-water; but as they approach it, the big streams grow sluggish and form more or less distinct deltas. The main rivers, taken in order from north to south, are the Rohkan, the Siak, the Kampar, the Indragiri, the Jambi, and the Pelambang. The last is said to be navigable for more than 200 miles for large vessels. Each river-system has numerous tributaries.

The climate of East Sumatra is warm and moist. Rains abound from October to March; the remainder of the year is considerably dryer. Although spoken of as "the dry season," it is far from arid. Thus at Kotta Ranah, lying inland in the Siak district, there were from the 15th of last June to the 1st of July three rainy days, on which the rainfall amounted to 71 mm.; during July it rained on 17 different days, the total rainfall amounting to 289 mm.; in August rain fell on 21 days to the amount of 282 mm., and during the first seven days of September rain fell on six days to the amount of 98 mm. That is to say, out of a total of 84 days in the height of the "dry season" it rained on 47 days, a total of 740 mm., or an average of 15.4 mm. for each rainy day. The rainfall at Hilversum and Merboenkahl, Siak, was, according to the *Natuurkundige Tijdschrift*, as follows:

	Hilversum, 1887.		Merboenkahl, 1888.	
	Rainfall, Mm.	No. of Days on which it Rained.	Rainfall, Mm.	No. of Days on which it Rained.
January	661	23	No observation.	No observation.
February	189	14	"	"
March	384	15	246	20
April	295	15	247	20
May	136	10	158	14
June	131	7	143	7
July	133	6	106	7
August	232	15	171	11
September	148	9	367	15
October	462	19	276	13
November	144	16	288	15
December	187	10	291	14
Total	3102	159	2293	136

That is, on an average it rained nearly every other day, with an average rainfall of over 18 mm. per day's rain. This is a better showing than that of Kotta Ranah, further inland, as given above; but the average would be greater if January and February had been recorded in 1888.

As a result of rains and the level character of the wide eastern country, flooding of the districts traversed by the main streams and their tributaries is a frequent occurrence, and marshy land abounds. This is especially the case along the Siak river and south of it.

The tin-fields of Sumatra are approached by steamers of 12 feet draught, running from Singapore, across the straits of Malacca to Brewer Straits, up the Siak river to Benkalis, thence south to Siak and east to Packanbaroe. The distance up the river to Packanbaroe is, according to the captain of one of the steamers, 120 miles. Beyond Packenbaroe the Siak river becomes very sinuous, is obstructed by fallen trees and logs, and, still further up, grows shallow. In consequence, all travelling from Packanbaroe up-stream is done in *bloncongs*, or partially covered canoes of a similar type to that used on some streams of South America (Magdalena river, etc.). The bloncongs are propelled by poling or rowing, as the case may require.

From Packanbaroe the Siak winds in an easterly direction to Getti, which lies about 20 miles along the river beyond Batoe Gadja, the place mentioned in the description of the boundary of the concession, as the one through which the north and south line runs.

There may be counted 176 curves or bends of the stream between Kwalla and Batoe Gadja. Kwalla, at the junction of the Tapong Kanau (right arm) and Tapong Kiri (left arm) forming the Siak river, lies about 20 to 22 miles upstream from Packanbaroe. This gives more than two curves for every mile along the river. Many of these bends are rectangular turns and quite a number of loops and return-loops are met, which make navigation impossible for any but small craft.

The cost of transportation—part by steamboat, part by canoe, entailing the breaking of bulk, the use of storehouses, etc.—is naturally a heavy charge against any enterprise commenced in this district. Whoever is acquainted with tropical streams, subject to floods and incidental dangers from drift-timbers, etc., will view the making of this river navigable, as a difficult and costly undertaking, especially if it is considered that the waters of the stream (more than 100

feet wide) rose at Batoe Gadja, in the "dry season"—between July 18th and September 13, 1890—more than 18 feet; yet this was an ordinary season. The substitution of land-transportation, for the natural waterways, would encounter special difficulties, arising from the marshy condition of that part of East Sumatra.

Any enterprise undertaken near the interior of Sumatra, is for the present confronted with inherent obstacles and disadvantages, involving heavy initial costs which can be offset only by extraordinary richness of the natural resources to be developed.

The present center of operation in this tin-field is Kotta Ranah, lying about 300 feet above the sea in longitude 100° 43′ East, latitude 0° 26′ North, and connected, by a trail not more than 15 miles long, with the Siak river-station Getti. Over this trail all freight is carried on the backs of the native Malays. The trail is bad. Probably one-third of it passes through marshes and swamps, over fallen trees and logs, where leeches, the pest of these regions, abound. The second third passes over low flat lands, and the remainder over rolling hills. Transportation over this distance cost formerly $1, Mexican money,* per load of 40 pounds; it has now been reduced to 66 cents per load. *Bloncong*-hire from Packanbaroe to Getti, for shipment of supplies, costs $30 to $35 per bloncong for the round trip; but if steadily employed, bloncongs can be rented for $55, Mexican money, per month. Communication is naturally slow. The average time occupied by a bloncong in a round trip, from the steamer-discharge pier in Packanbaroe to Getti and Kotta Ranah and back, is 15 days. Temporary heavy floods or very low water may increase the time required; as much as 22 days have been consumed in the trip.

I have emphasized the drawbacks and inaccessibility of this region, in order to contrast it with the naturally advantageous situation of Banca, Billiton and other tin-producing competitors.

Kotta Ranah lies about 6 miles northeast of Bukit Siligi, the three high peaks in the most easterly chain of the divisional ranges. The axis of this range runs northwest, and southeast, and mountain spurs extend from it, to the northeast out into the lowlands. These spurs form local water-sheds to the tributaries and sub-tributaries of the Siak and other rivers. Kotta Ranah is a native hamlet (*Kampong*) situated on the stream (*Sungi*) of Lau. The Sungi Lau unites beyond Kampong Kebon with the Pelambayan, forming the Talanca, a tributary to the Tapong Kiri (the left arm of the Siak river).

* The *peso* or Mexican dollar is current in Sumatra and Singapore as the trade-dollar of the country.

I may here note that the Malay names the banks of a river as he faces them in going up-stream, and calls that the left bank or left arm which we would call the right.

The nearest *Kampong* to Kotta Ranah is Rambei, on the Sungi Rambei. It lies about two miles, in an air-line, northwest from Kotta Ranah. The two hamlets are separated by the foot-hills of a narrow side-spur, divide or rib, which forces the Rambei stream northeasterly to its junction with the Getti river. The latter has its confluence with the Tapong Kiri (left arm of Siak river) at or about the settlement of Getti.

The Rambei and Lau are two distinct stream-systems, but both are sub-tributaries to the Tapong Kiri. The two systems traverse a considerable area, when it is considered that the streams extend yet for miles beyond Kotta Ranah and Rambei, and that the distance from Kotta Ranah to Batoe Gadja, via Kebon and Pelambayan, is nearly 19 miles by measurement; from Kotta Ranah to Getti, estimated, say 14 miles, while Getti and Batoe Gadja are probably only 10 to 12 miles apart in a straight line.

Any kind of material entering the creeks or by-creeks of these streams, is likely to be carried through the territory and eventually into Siak river.

I may appear to be entering into tedious detail in this matter; but the water-courses of a country are the agents of the distribution of tin-ore, which may there be found widely distributed as stream-tin, far from its source. Moreover, this country has been visited by few white men; it is to a large extent unsurveyed, and it is incorrectly represented on existing maps. The record of my own observations may therefore be valuable to future travellers.

I have indicated the relative position of Rambei and Kotta Ranah in Figs. 1 and 2, made from data personally obtained, in a rough compass-reconnaissance. Fig. 2 is the southeast continuation of Fig. 1, but has been engraved for clearness upon a scale twice as large.

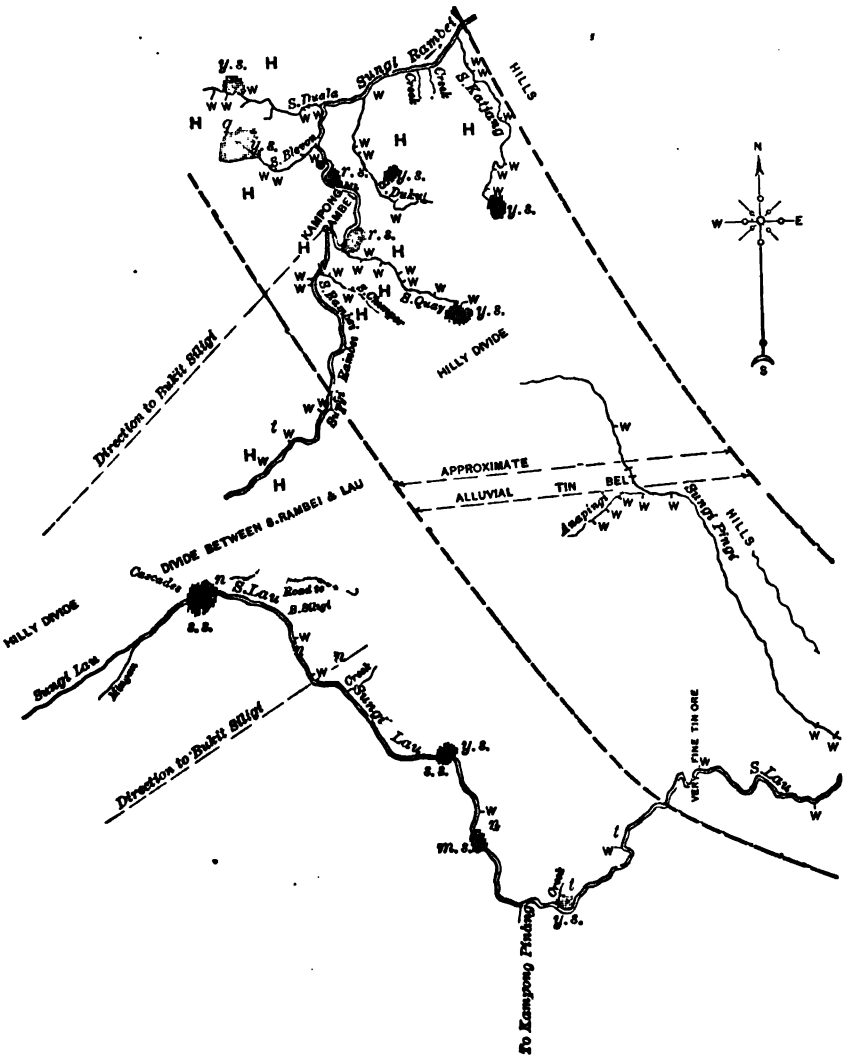
The Lau-Rambei system of streams with its tributary creeks, forms the basis for the tin-operations of the concession above described.

Though I traced the waters of the two main streams further than the sketch shows, I have limited the representation to the actual compass-reconnaisance.

A careful examination of the only natural exposures of rock, namely, those occurring in the water-channels, and local tests made

FIG. 1.

SKETCH MAP
of
THE RIVERS LAU AND RAMBEI
WITH THEIR TRIBUTARIES.
Siak, Sumatra.
Scale 8000 Ft. to the Inch.



H, H, hills. Rock-exposures are shaded and marked: *r.s.*, red sandstone; *y.s.*, yellow sandstone; *m.s.*, micaceous sandstone; *s.s.*, siliceous sandstone; *q.*, quartzite. W,W, points where ground was tested by washing for tin-ore; *t.*, traces of tin-ore; *n.*, no tin-ore.

by borings and pits, leads me to conclude that the tract under consideration is everywhere underlaid with impure sandstones and quartzites. The partially decomposed red and yellow sandstones near Kampong Kotta Ranah become reddish-gray and grow micaceous and siliceous, as they are traced in the direction of the head-waters of the Sungi Lau, towards Bukit Siligi. In the lowlands of Kotta Ranah, the dip is generally very slight to the southeast. As the hills are entered the dip increases, and locally, along the upper Lau (first cascades), the sandstones are tilted vertically, though flattening again beyond.

This hard, fine-grained, gray Cascade sandstone consists, according to miscroscopic examination, of rolled grains of quartz, with abundant amphibole and biotite, scales of white hydro-mica, particles of kaolinized feldspar and iron ocher.

The head-waters of the Sungi Blevon (Rambei district) come, to judge by appearances and their mode of occurrence, from a fine-grained reddish or greenish quartzite.

Near and west of the headwaters of the Sungi Batang (Kotta Ranah district), and for 1500 to 1700 feet along its banks in the ravine, boulders, blocks and pieces of a black compact quartzite, somewhat resembling the whinstone of Cornwall, were found, indicating its occurrence *in situ* on the adjacent hill.

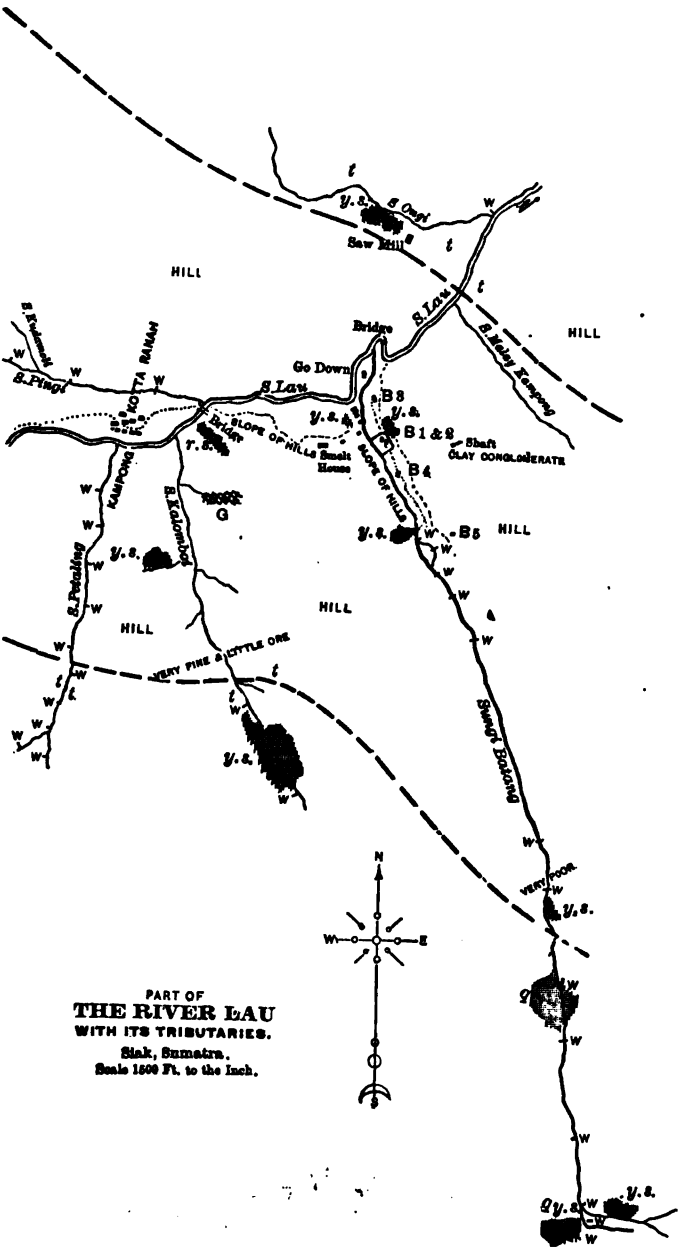
The two quartzites differ in appearance, but under the microscope they seem to show similar origin and constituents. In fact, the transitions to the siliceous and micaceous sandstone, so well marked to the eye, nearly disappear under the microscope. Even the ordinary red, yellow and yellowish-green impure sandstones show similar constituents, only in differing proportions and varying stages of decomposition. As a check to my own microscopic examinations of finely pulverized rock-samples, I submitted different characteristic samples to Dr. A. Julien, of the Columbia School of Mines, for thin-section examination. What I term quartzites Dr. Julien calls "sandstones, quartzitic in hardness and compactness," otherwise we agree. All the rocks are unquestionably of sedimentary origin.

The quartzite of Sungi Blevon is composed of rolled grains of quartz, with an abundance of amphibole, a little biotite and menaccanite.

The quartzite of Sungi Batang has the same constituents, only amphibole occurs in very small proportions, and a small quantity of pyrite is present.

The yellow and red sandstone represents results of decomposition.

FIG. 2.



For explanation of marks, see Fig. 1; also, G, exposure of granite; B, B, bore-holes. Bore-holes B 1 and 2 are in the *kulit*, or main working.

The more or less rounded grains of quartz, which predominate, are cemented by iron ocher, in association with a fine mixture of decomposed scales of hydro-mica and particles of biotite, amphibole, feldspar and menaccanite.

Besides the sedimentary rocks mentioned, there is a place near the red sandstone exposure, east of Kampong Kotta Ranah, where granite has been laid bare by the last right-hand tributary of the Sungi Kalomboi, for a distance of 60 to 75 feet.

This granite shows, in a quartzose and albitic matrix, white muscovite and black tourmaline. Within the area exposed is a lode about 6 feet wide, of a very fine-grained granite, contrasting with the coarse inclosing granite, from which it is separated in addition by a smooth plane. It appears like a product of segregation. The composition of the two granites seems similar, but the proportion of quartz is larger in the lode. Bedding-planes are observed, suggesting what Dr. Hunt calls an indigenous granite.

The Kalomboi streams show more or less abundant traces of this rock up to the next tributary on the right, suggesting a similar exposure in the next ravine. No tin could be found in this granite. South of this and along the west banks of the Kalomboi, the yellow sandstones are observable.

Fragments of a similar granite were found in the Blevon and Katjang creek-beds, which would indicate that a granite exists north of the Lau-Rambei divide, though diligent search failed to find it in place.

The density of vegetation makes prospecting difficult. Almost every line explored has to be previously cleared by bush-men. In view also of the limited time at the disposal of a transient explorer, the interruptions and delays caused by frequent drenching from tropical rains, and the dissuasion from too much minute examination of the jungle furnished by occasional tiger-tracks, it will be clear that a thorough study of the geology of the region was not practicable for me. Such a study will require long and systematic work. Few companies have money to spare for that.

Briefly summarized, the red sandstones crop out southeast of the Sungi Pingir and Lau confluence, and again west of Kampong Rambei, toward the Bleoon-Rambei confluence, and south from Kampong Rambei, towards Sungi Quay. All sandstone exposures found (beyond those enumerated) were a yellow sandstone, more or less decomposed by weathering, save along the upper Lau, where

the indurated, reddish-gray, micaceous and siliceous sandstones occur.

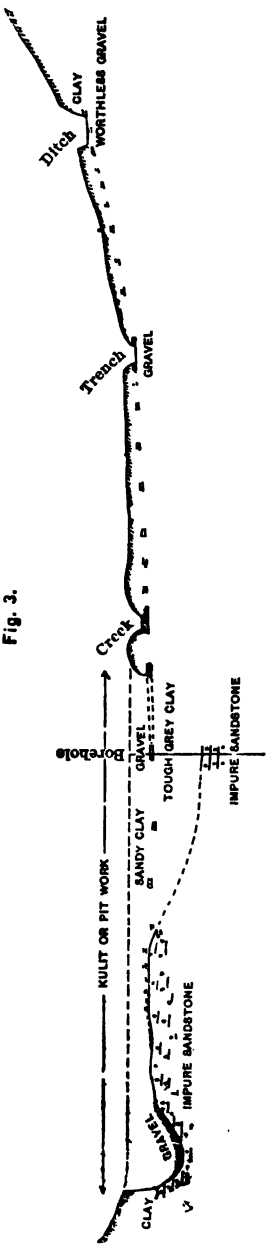
It is possible that the albitic granite more generally underlies the sandstone beds of this district than is proved by the data now at hand. But it is certain that granite does not come to the surface, for no mica is found in the beds of the mountain streams.

The country north of Kotta Ranah, towards Getti, is slightly rolling for about 6 miles, the remaining 8 miles being half lowlands, half marshes and swamps. The highest altitude, according to corrected readings from a compensated aneroid, is 560 feet above the sea, and the lowest hilltop is 300 feet high, while the lowlands range from 200 feet above the sea-level down to 115 feet (at Getti). The underlying formation of this north country, so far as can be judged from the color and nature of the soil on the paths, is sandstone. At and east of Getti an indurated calcareous clay full of *cypridenas* and other small shells, forms the shores of the Tapong Kiri. At Batoe Gadja, about 20 feet lower, the banks and the bed of the river are indurated fine clay-conglomerate. The clay matrix surrounds rounded quartz granules, from the size of a pin-head to pea-size, and more angular and partly kaolinized pieces of feldspar. Throughout the mass occur carbonized vegetable remains, and an imperfect impression of a shell resembling *area* was noticed. East of Petapahan, along the lower Tapong Kiri, sandstones occur of more recent formation than the (probably Tertiary) Kotta Ranah series. Inland, up and down the Siak river and west of the town of Siak, the soil is very clayey, so much so in fact that tobacco-growing has been abandoned on account of it, as I was told by one of the former tobacco-growers.

A comparison of the foregoing account of Sumatra with what is known of adjacent countries, shows that similar geological conditions have prevailed in the Malay Peninsula, Karimon Islands, Western Borneo, Singkep, Banca, and Billiton, and as the search is extended, this belt of a former geological unity will unquestionably be widened. Quartzites, quartz-conglomerates, quartz-schists, hornblende-schists, siliceous sandstones, clay-slates, with granite and local basalts (Karimon Islands, according to *Natuurkimdig Tijdschrift von Nederlandsch-Indie*, Vol. XLIV.) form the prevailing rock-formations.

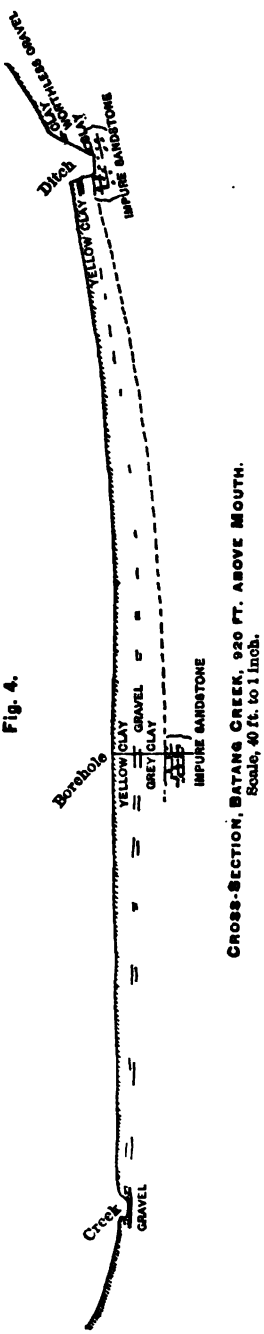
In examing the Kotta Ranah-Rambei stream-system and banks for alluvial tin, I found a universally shallow and similar alluvial covering. Figs. 3 and 4 present two typical sections, made at

Fig. 3.



CROSS-SECTION, BATANG CREEK, 550 FT ABOVE MOUTH.
Scale, 40 ft. to 1 inch

Fig. 4.



CROSS-SECTION, BATANG CREEK, 920 FT. ABOVE MOUTH.
Scale, 40 ft. to 1 inch.

local widenings of the ravine, one 550 feet, and the other 1470 feet from the confluence of the Batang creek with the Sungi Lau. They show the increasing shallowness of the alluvium found in going up stream, which continues until the bedrock appears in the creek-bed, the ravine contracting in the meanwhile to a final width of from 15 to 35 feet.

The covering over the angular quartz-gravel averages 3½ to 4 feet, and consists of about 6 inches of humus and 3 to 4 feet of yellow sandy clay. The underlying angular quartz-gravel is itself divided into two separate layers. The upper, 8 to 10 inches, consists of angular fragments of white quartz, of 3 inches maximum, and 1 to 1½ inch average diameter, largely intermixed with carbonized wood, and contains occasionally stray pieces of fossil *damar* resin. It carries very small amounts of fine cassiterite and considerable magnetic and red oxide iron-sand. In the lower layer, which is separated from the upper by a thin, often only slightly marked, seam of gray clay, from 1 to 2 inches in thickness, the nature of the quartz-gravel is the same as above; but carbonized wood is absent, the amount of iron-sand is smaller, and that of cassiterite is larger. This layer is locally known as the pay-seam. Small crystals or crystalline fragments of spinel and ruby, but no tourmaline, garnets, topaz, or mica, were noticed. The cassiterite is to a large extent semi-crystalline, and crystals are by no means rare. Both the tin-ore and the quartz show little average wear of corners and edges, indicating that they have been transported but a short distance only.

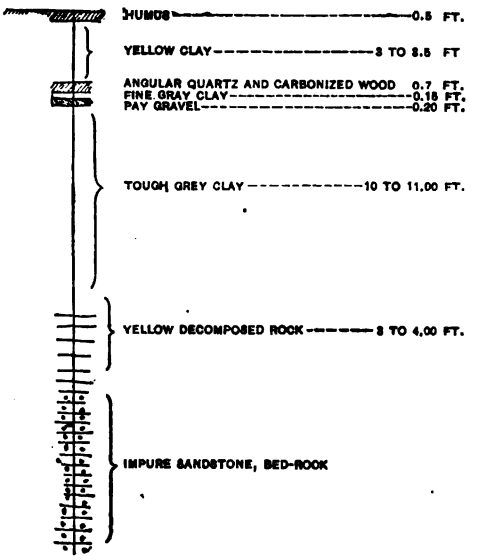
Underlying the quartz-gravel is a tough gray or greenish-gray clay of varying thickness, which gradually passes into decomposed rock in place from 3 to 4 feet thick. Below this lies the true bed-rock, an impure sandstone of the nature already described. Fig. 5 illustrates the above description.

The tough gray clay underlying the quartz-gravel is the bed-rock for working purposes. As is usual in creek-beds and their adjacent banks, unevenness in the bed or pot-holes occur, in which there is generally an extra accumulation of tin-ore. In figuring the thickness of the gravel and its corresponding richness they must be considered. By a simple computation (the tons of quartz-gravel worked from a given area being known) I ascertained that the average thickness of the pay-quartz-gravel, including the pot-holes, is about 6½ inches, or 0.54 feet in the Batang creek, with an average overburden of 4.93 feet.

This creek had been selected to begin operations on, as offering the best inducements.

The pay-gravel yielded "black tin" (averaging 70 to 72 per cent. in "white" or metallic tin) at the rate of $2\frac{7}{10}$ pounds per ton of 2240 pounds or 0.12 per cent. Calculated on the total amount of ground excavated, including the stripping, this would be 0.476 pounds black, or 0.348 pounds white tin per cubic meter excavated. In Banca, according to Van Diest, the same amount of excavation

Fig. 5.


HUMUS 0.5 FT.
YELLOW CLAY 3 TO 8.5 FT
ANGULAR QUARTZ AND CARBONIZED WOOD 0.7 FT.
FINE GRAY CLAY 0.18 FT.
PAY GRAVEL 0.20 FT.
TOUGH GREY CLAY 10 TO 11.00 FT.
YELLOW DECOMPOSED ROCK 3 TO 4.00 FT.
IMPURE SANDSTONE, BED-ROCK

AVERAGE ALLUVIAL DEPOSITION, AS PROVED BY THREE INCH BORE HOLES KOTTA RANAH, SIAK DISTRICT, SUMATRA.
Scale, 10 ft. to 1 inch.

yields $2\frac{2}{10}$ to $3\frac{1}{3}$ kg. black, or $1\frac{3}{4}$ to 2.03 kg. (2.95 to 4 46 lbs.) white tin.

The relative proportion of pay-gravel to entire stratum removed is at Banca, 3 to 33 feet; at Kotta Ranah, 0.54 feet to 5.47 feet, or practically the same. The advantage of a lighter covering in the latter place is neutralized by a reduction of yield in about the same proportion. The economical limit of mining is naturally different in the two places, depending largely on surrounding conditions, mechanical facilities and the cost and efficiency of labor.

In Banca, the amount of ground removed per annum by one man is, according to Van Diest, 300 to 450 cubic meters. At Kotta

Ranah it is 205 cubic meters in cleared, dry ground, and 196.68 in cleared but ordinary ground. The average yearly product of one man in Banca is 610 kg. white tin.

Hydraulic stripping cannot be introduced at Kotta Ranah for lack of head and grade. The maximum width of the pay-channel is about 200 feet; the average probably 35 feet. The streams are short; and if mechanical devices were used in stripping, they would have to be moved frequently.

Pannings were made of nearly 10,000 pounds of gravel from the different creeks. Twenty per cent. of the pannings were rejected as too low, or as lying outside the limits of the tin-bearing channel; the remaining 80 per cent. gave an average result of 1.13 pounds metallic tin to 2240 pounds of stream-gravel, or 1.75 pounds per cubic meter.

The distribution of the cassiterite through the creek-beds of this district is somewhat unusual in character. One would expect the fine ore to be found down-stream and the coarse ore up-stream, according to the laws governing the deposition of sediments. But in this examination I found, on the contrary, a perceptible decrease in the average size of the grains, going up-stream, within a certain limit or belt. Going beyond this limit there was a positive decrease, both in the coarseness and quantity of the tin-grains. I also noticed an absolute unsized condition, so to speak, of the tin-grains in the creek-beds, within this belt or zone, while outside of it the tin would either be extremely fine and but little of it, or no tin at all would occur. (Upper Lau and Rambei). Within the belt coarse tin-ore would be found in the creek-beds, a few feet beyond fine ore, and *vice versa;* there seemed to be an utter absence of regular sizing.

In some of the streams, notably the Petaling, Kalomboi, middle Quay and upper Blevon, grains or crystals of cassiterite were found still clinging to small, angular, or sharp pieces of quartz, which appeared crystalline and drusy under the microscope. This indicates that the tin-ore now found in the streams has been broken down from veins of quartz not far away. In some instances (Petaling and Blevon) the veins must be very near.

Considering also the angularity and size of the quartz-gravel in the creek-beds and banks, and the absence of mica and tourmaline, one is inclined to think the existence in the indurated sandstones of narrow quartz-veins carrying cassiterite, which have furnished the stream-tin, to be a not unreasonable hypothesis.

By plotting the results of the test-pannings, a tin-belt is defined

as I have indicated in the accompanying maps, with a northwest and southeast trend. This indicates that the course of the stanniferous quartz-veins (from which the stream-tin came) corresponds with the general axis of upheaval of the country. The nature and unsized condition of the detrital material points to an occurrence of these veins in the sandstones lying but a short distance above the marked belt, or even within it. Corroboration bordering on proof is furnished by the absence of quartz-gravel in the upper Lau (towards its headwaters), and the occurrence, instead, of micaceous and fissile sandstone in pieces of rounded and angular shape. Similar, but less marked conditions occur in the upper Rambei stream, while in the upper Batang and other streams all gravel disappears.

It thus appears that the Kotta Ranah-Rambei alluvial tin-field is of very recent origin, being derived from the broken-down outcrops of narrow stanniferous quartz-veins, which occur in the underlying and adjacent impure sandstone, in a probably northwest and southeast direction. They are likely to extend at intervals into adjoining districts.

That there really are narrow quartz-veins in the underlying sandstones (though what I saw was not stanniferous), I had occasion to satisfy myself when a pit was sunk in the *kulit* of Batang creek, extending into the decomposed rock in place. A nearly vertical two-inch veinlet of quartz was carried in one side of it.

The quartz now found in the creek-beds is of course part of the broken-down outcrops of veins. The surrounding country indicates that erosion has taken place, to a minimum depth of several hundred feet. The heavy tropical floods carry as a matter of fact the finer and lighter tin-ore to the larger streams and their flood-plains; and it is not surprising that traces of tin-ore can be found even at long distances from their probable source. Unquestionably there exist other and similar belts in Siak. I heard of tin-ore in the streams of the northwest part of the concession and beyond, near to Rohkan river; but I did not go to test their value and extent. A systematic further tracing of the belt may lead to the discovery of richer fields in Siak than the one described.

It may not be amiss to introduce for comparison's sake, some figures from another foreign field, the Kong Loon Kongsi mine, on the Malay peninsula, in the district of Kamounting, Perak, as reported by Mr. M. J. Errington De la Croix in *Les mines d'étain de Pérak*. A Chinese company is working the field under a concession, and the formation may be taken as typical for that district.

At this place, a cross-section through the alluvium shows

	Meters.	
Humus,	0.20	Strippings, 3.5 meters.
Yellow clay,	0 90	
Sand with tourmaline grains,	0.10	
Gray clay,	0.65	
Gravel, quartz, feldspar, tourmaline,	0.25	
Reddish ferruginous clay,	0.50	
Sand,	0.15	
Yellow clay,	0.75	
Tin deposit with bowlders,	2.80	Pay gravel, 2.8 meters.
Kasolin (bed-rock),	?	

The pay-gravel bed averages 1 per cent. in metallic tin. From Mr. De la Croix's figures it appears that 44.4 per cent. of the material excavated is useful material and the remaining 55.6 per cent. is waste. He says it requires 41 cu. m. (at 1600 kg.) of pay-gravel to make 1 ton of black tin-ore, assaying 66 per cent. white metal. Hence it must require 92.48 cu. m. of excavation from grass-roots to bed-rock to furnish the 41 cu. m. of gravel required. This costs $1.18 (Mexican money) per cubic meter of gravel. (The book, page 36, says 95,202 cu. m. and $1.15 cost; a slight error).

From other data in the report, I calculate that one man removes in that mine 343 cu. m. of ground per year. The work performed at the Kong-Loon mine, is raising 16 kg. (the weight of a basketful of gravel) a maximum of 6.30 inches, with a velocity of 5 centimeters per second. The day's work amounts to about 6½ hours. The work performed is then 0.80 kilogrammeters per second, or 18,720 per day. At Kotta Ranah, the workmen raise only (average of 56 observations) 11.6 kg. (25.59 lbs.), a maximum of 1.72 meters, with an average velocity of 3 centimeters per second, or the duty performed is 0.348 kilogrammeters per second, or per day's work of 7⅞ hours, 9605 kilogrammeters.

The class of workmen and the discipline and organization are evidently superior at Perak; but the duty performed, even there, is only about one-third of that of white labor in temperate climates.

In view of this large difference in effective work, a few words as to the system of mining labor in the East may be placed here.

The ordinary workmen employed for mining and allied occupations are Chinese, who are either imported direct, by the employers, or are obtained more generally through Chinese agencies, which exist in different parts of the East. These agencies import the men direct from China, at their own expense, under contract, and then

sub-let them to individuals or corporations, at a fixed cash price per head (supposed to cover expenses and profit of the importing agencies) and under agreed and stipulated conditions with their employers, which are signed by both parties and attested in the presence of the "Chinese protector."* The wages of the ordinary coolie are $30 Mexican money, per year, $18 of which is advanced to him upon enrollment. The agency in Singapore receives in addition a cash payment of $20 for a 12-months' coolie. At the termination of his contract, the laborer becomes an "old or free coolie" and is at liberty to re-engage of his own volition, at a previously agreed price, to the same party, or make his own terms elsewhere, or return to the town whence he came. As a novelty to American miners, I give a copy of one of these contract-forms.

General Form of Contract.

For Chinese Engaging to labor in this Colony and in the Native States.

It is this day mutually agreed between the employer the A. B. C. Company, his attorney, heirs, or assigns, and Tu-Tru, Chinese laborer, born at ———, and aged — years:

That the said laborer is willing to proceed to Kotarana Sumatra and be employed there as a tin-miner, for twelve months, at a yearly wages of thirty dollars subject to the following conditions, viz:

1. That the said laborer receives an advance of eighteen dollars, which shall be deducted by instalments by the employer, at the rate of $1.50 per month.

2. That the expenses for conveying the laborer to his destination shall be borne by the employer.

3. That the said employer shall furnish the laborer with a suitable house, for which the latter will not be required to pay rent.

4. That the said employer shall provide the laborer with his daily food; and also furnish him with one jacket, two pairs short trousers, one mosquito-curtain, two bathing cloths, one sun-hat, and a pair of clogs.

5. In the event of the laborer falling ill from natural causes, the exployer shall furnish him with medicine and a place for his medical treatment until recovery; and if the days of illness do not exceed thirty (30) days, the loss of time shall be borne by the employer,

* The Chinese protectors are Government officers, appointed to enforce the fulfilment of labor-contracts and to take the part of the wronged party, if occasion requires.

and the laborer will not be required to make up for it. But should the illness of the laborer exceed thirty (30) days during one year, or should he fall sick from his own fault, or contract any venereal disease, he shall, on recovery, or after the termination of his agreement, make good the days of his illness; and shall also pay to the said employer twenty-seven cents as cost of food for each day's absence. Should the laborer desert and be captured, all expenses actually incurred shall be repaid by him.

6. Should the laborer be unable to work on account of venereal disease, or stop work through laziness, the number of days of such absence, together with any advances he may have received, shall be endorsed on the contract; and should there be, at the expiration of the agreement, any sums outstanding, the laborer shall work at the rate of six dollars per mensem, as in the case of old coolies, until the whole amount has been paid.

Whenever there is any dispute, the agreement shall be taken to the Protector of Chinese, or, if beyond this Colony, to the local authority.

7. Ten hours shall constitute a day's work, but in case of emergency the laborer shall work beyond the specified time. Such overtime shall be placed to the credit of the laborer at the rate of wages mentioned in his contract.

8. The customary Chinese festivals will be considered as holidays.

The above eight (8) articles having been clearly explained to both parties by the Protector of Chinese, they have agreed to all of them and have signed this contract with the understanding that they shall hereafter observe all the articles mentioned therein.

Register No.—; Name in English of employé, —— ——; Age —; Name in Chinese, —— ——; Original country ———; Advance $——; Signature or mark of employé —— ——.

Office of Protector of Chinese, ——, 18—,

—— —— *Employer.*

—— —— *Witness.*

—— —— *Protector of Chinese.*

Coolies introduced into the Dutch possessions must have their contracts filed and registered by their employers, with the nearest

"Resident" or "Assistant Resident." The Dutch law does not permit coolies to be worked without a contract, and requires a proper enforcement of its stipulations. So long as coolies live up to their contract they cannot be dismissed before its expiration, without the consent of the "Resident." This is an important matter to the promoters, in case of an unprofitable enterprise. Coolies work under foremen called *Mandors*, and distinguished as first, second, third or fourth mandor, according to their ability and duration of service, and according to the number of men under them. In Siak each mandor worked 50 coolies. A mandor's wages vary from $100 down to $70, $60, or $40 (Mexican money) per month. The mandors are supposed to board themselves; they are paid from one to two month's wages on account, on enrollment. Mandors are picked from among "old or free" coolies. The wages of Chinese blacksmiths (free), vary from $20 to $25 per month, and carpenters cost, according to skill, from $15 to $25 per month.

To these wages must be added the cost of brokerage, agency, contract-drawing, medical examination and other items; also the cost of shipping coolies to their destination and feeding them *en route*. The monthly board, either provided by the employer direct or through a boarding-house boss (*Tauki*), against a fixed charge, must also be included. If a coolie dies during contract-time, the $20 purchase-money and any part of the $18 advance not yet worked out, is lost to the employer. Moreover, the time lost through sickness of less than 30 days, the cost of medicine administered (see contract), the hospital costs for patients ill more than 30 days and the eventual funeral expenses and loss of clothing supplied, are all charges to be considered when the cost of a unit of labor is computed. The time lost during heavy rains (the coolies working out of doors), when calculated for a lot of coolies per year, represents a good many days work. Adding to all this the necessity of breaking in raw coolies for their especial work, we may conclude that coolie labor is more expensive than we might at first have supposed.

The hours of work in Siak are from 7 to 11 A.M. and from 1 to 5 P.M. About ten minutes are lost each time on going to work and getting ready; the coolie's work is reckoned to commence when he leaves his roof or shelter.

Coolies in Siak are paid once a month, nominally $2.50, but $1.50 is kept back on account of the advance made at enrollment, leaving the coolie $1 in cash per month during his year of employment. There are no rates per hour, such as are established for over-time in the

Malay peninsula, where the whole coolie labor-system, being of longer standing, is better organized and disciplined than in Siak.

The tools of the Chinamen are a *chankol*, shown in Fig. 6 (a hoe-like instrument, which takes the place of the pick and shovel), and a crow-bar, for harder material or boulders. The material excavated is filled with the *chankol* into small, flat rattan baskets

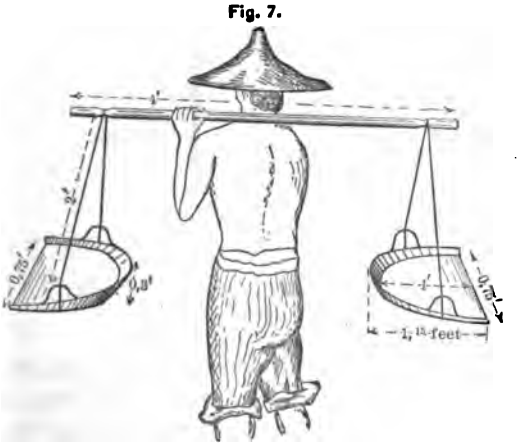
Fig. 6.

MINER'S CHANKOL.

(*bakol*), which are carried across the shoulders, as shown in Fig. 7, being hung back to back, so as to be easily dumped by either hand. Notched poles, similar to those used in Mexico, serve as ladders in the pits.

To raise small quantities of water, or to hoist from the deeper pits, the old-fashioned " well-sweep " is used, with a thin rattan vine

Fig. 7.



COOLIE CARRYING BAKOL.
¼th Natural Size.

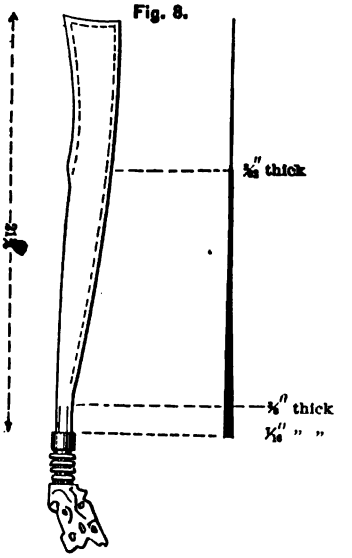
for a cord, carrying for the former purpose a bucket, for the latter a larger *bakol* than is used for carrying dirt and gravel by hand. The Chinese pump is also employed. Steam-pumps have been introduced in the Malay peninsula.

In addition to the Chinese miners, small numbers of Javanese coolies are employed in Siak for making roads and grubbing out the roots of trees, and the native Malays find work in clearing the jungle

and forest, carrying supplies over the rough roads, building native huts, coolie-houses, stock-houses, etc. The Javanese coolies are, like the Chinese, under time-contracts, but they feed themselves, procuring supplies from the natives. The Malay workmen are free men, independent in their actions and character, and better suited to their special work than either of the other two races. The three races live apart, and there is no love lost among them.

Among the Malay tools the *parang* and *blevong* deserve mention. The *parang* (Figs. 8 and 9) is a sword-like knife, reminding one of the Spanish *macheta*, but heavier at the end. It is used for cutting

Fig. 8.



PLAIN BUSH-PARANG.
⅛th Natural Size.

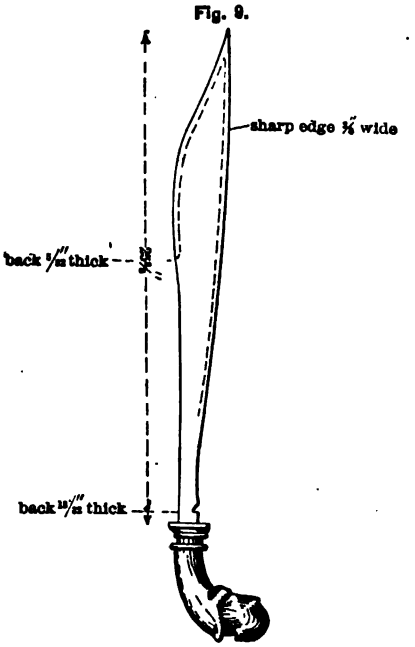
bushes and shrubs, and is a formidable weapon in skillful hands. The *blevong* (Fig. 10) is the native axe. The long, thin handle, slightly elastic, is set at 75° to the axis of the steel blade, which is attached to it by split rattan, firmly bound around both. The axe is held with both hands by the upper and thicker part of the handle, and is operated without apparent effort with a side-swing not higher than the shoulder, cutting, in skillful native hands, with surprising rapidity and accuracy.

Malays are paid 33⅓ cents a day, and feed themselves.

Storehouses, coolie-houses, etc., are framed of rough timbers and heavy sticks, the sides being of bamboo laths covered with bark or *atap* (a variety of palm-leaf). The floor is made of closely-laid

bamboo sticks, and the roof is covered with *atap*. The following example shows the number of days' work required to build, as far as the lower edge of the roof, a storehouse (*godang*) 14m. long, 8.5

Fig. 9.



BUSH AND DEFENCE-PARANG.
½ th Natural Size.

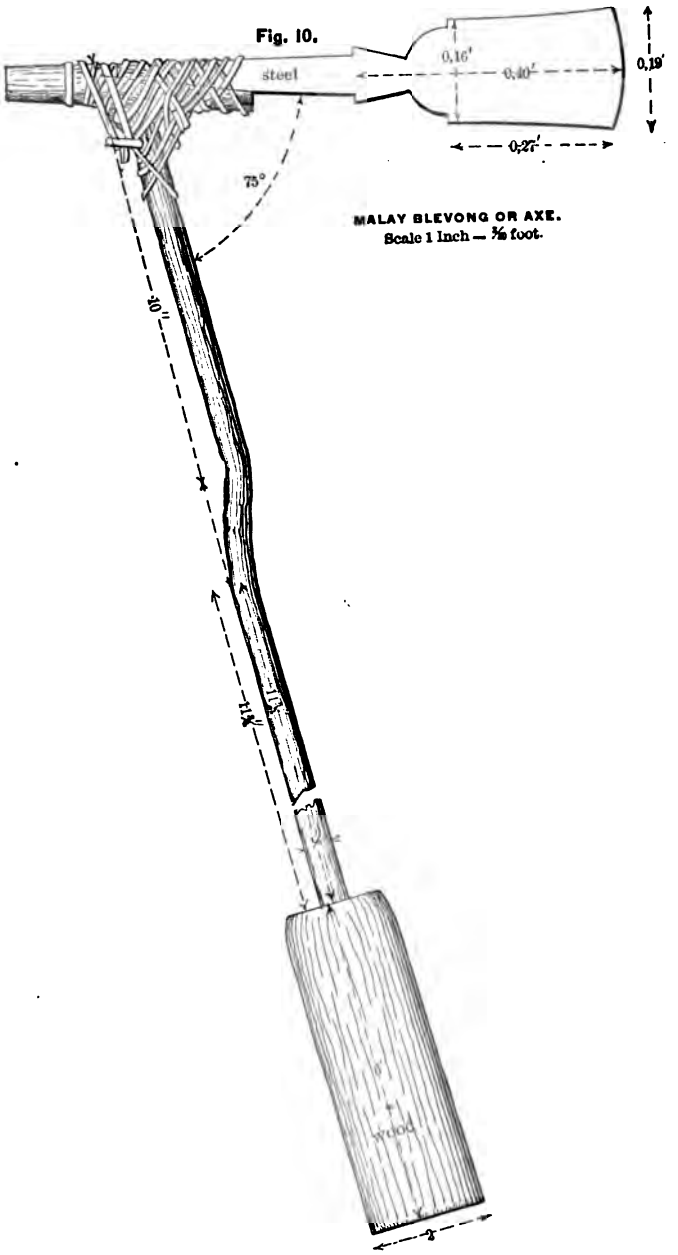
m. wide, and 2.7m. high, and raised on posts 1m. high from the ground. The jungle was cleared for a considerable distance around the house:

	Days' Work.
Clearing jungle,	24
Grading,	46
Cutting timbers and carrying to camp,	326
" sticks " " " "	49
" *atap* " " " "	350
" rattan-vine " " "	150
" bark " " " "	12
" bamboo " " " "	57
" extra wall-material, " "	41
Preparing *atap* in camp,	262
" rattan "	154
" wall-material in camp,	120
Labor of erection,	364

Total, 1955 days' work, at 33⅓c., = $651.66.

A dwelling-house 11 x 8.5 x 2.3 m., with kitchen attached, 5 x 5 x 2.5 m., cost 324 days' work. Both buildings were rain-proof.

Fig. 10.



MALAY BLEVONG OR AXE.
Scale 1 inch — ⅜ foot.

The Banca miners of alluvial tin distinguish two classes of work :

the mountain stream-work, called *kulit*, and the working of valley-placers and flats, called *kollong*. From this has also grown the custom of calling work on an upper gravel-layer, *kulit*, and on a lower one *kollong*; work on an intermediate layer being called *kulit-kollong*.

In Siak, Sumatra, *kulit*-work only exists. This work is performed, after the necessary preliminary cleaning and grubbing, by Chinese coolies, and consists in first diverting the stream and then making, stream-upwards, a series of rectangular excavations (*kulit*), taking in, if possible, the full width of the pay-channel or working, from bank to bank down to and through the pay-gravel, but keeping the strippings separate. After staking off a rectangle of the width of the gulch or pay-channel, and say a hundred feet in length, a number of Chinamen are placed, *chankol* in hand, along the lowest line across the gulch, facing stream upwards. These loosen the ground and fill it into the baskets of the stripping-carriers (*angkat tanna*), who stand behind the diggers (*chankol tanna*) and have to carry the waste back of the field to the dumping-ground. When the staked area has been stripped down to the pay-gravel, or *karang*, this is taken out similarly and carried to the ore, or *karang* floor.* A new pit is then commenced, and the strippings of the new work are dumped into the first excavated, and now empty, pit. When the work is in full operation there are distinct squads of strippers and pay-gravel diggers with their respective squads of carrier's distinguished as *orang tanna* and *orang karang*. The gravel is constantly tested, at different places, by washing (in the *batea*) to guide the diggers.

As already observed, the baskets at Kotta-Ranah in which the material is carried out of the pits are small, holding only 25 lbs.; in Perak the carrier's load is 35 lbs., and in Pahang, as I am told, 37 lbs. If light iron wheel-barrows were substituted, and planks laid to wheel on, a carrier, in these shallow deposits, could just as easily wheel out from 60 to 70 pounds in the time he now consumes in carrying 25 pounds. The Chinaman, however, dislikes any innovation upon his accustomed ways, and I encountered much prejudice on the part of the *mandors* when I suggested the trial. They were, however, finally persuaded, and I convinced them, watch in hand, that my assertion was correct. But I soon learned that, besides

* *Karang* really means "rock," and is used, in this connection, as our lode-miners speak of "pay-rock," in contradistinction to *tanna*, or earth. *Angkat* means carrying or carried.

prejudice, superstition had to be overcome with the Chinese tin-miner. The tales of mining kobolds and gnomes still hold full sway in their minds. They believe that tin is under the especial protection of gnomes who are easily offended, and who, if established ways are not strictly adhered to, leave the mine and take the tin with them. One very disagreeable thing exacted by these gnomes is that when *karang* is being dug, no one shall enter the *kulit* except *barefooted*. Now this is the most interesting stage in the work for one who wants to study the deposit; and it surely tries an expert's enthusiasm to be obliged to walk barefooted over muddy, angular quartz-gravel, for each sample to be panned. But I had to do it. Again, while pay-gravel was being dug, I was not permitted to approach the ore-floor or the *kulit* with an opened umbrella, to shield myself against rain or the tropical sun. The *mandor* told me that if I did so, "the tin would leave the gravel over-night."

At each starting of a new *kulit*, or of the wash-machine, tapers were burned at each end, formulæ were said in Chinese, and the wash-machine was treated to a handful of rice, carefully dropped into the feed-water. It required patience and time to overcome these superstitions. Though impressed by my washing tin out of the gravel in a gold-miner's pan, the coolies did not give me their confidence until I brought out and started a prospecting borer, which, to their surprise, brought up tin, without stripping in the regular way. I was permitted after this to keep my shoes on, but I still had to forego the luxury of an umbrella about the diggings.

Another evidence of either superstition or extraordinary mutual affection is presented by the death-register. Each coolie receives a number on enrollment which is carried for identification; in addition each man is photographed, and his number is written on the picture. Friends generally take consecutive numbers. I was struck to see how death selected certain sets, nearly by consecutive numbers, such as: 9322, 9324, 9325, 9326, 9328, 9344, 9346, 9347, 12,513, 12,515, 12,517, 12,519, 12,520, 12,541, 12,542, 12,543.

Can this be accidental, or is it a proof of the physical effects of sympathy, or of superstition?

The simple and crude borer used in the East for prospecting alluvial deposits, consists of a circular iron platform, with a central annular opening, and a projecting sleeve on the under-side large enough to slip over the casing-tube, to which it is held by three heavy screw-bolts. The casing has couplings with slot key-seats or common screw-couplings. The pipe used was 3½-inch cast-iron, with a pro-

jecting steel shoe for the end-piece. Boring-rods were 1½-inch square bar-iron, with a screw-socket top. The bit of the boring-tools is 3 inches in diameter; there is a variety of bits in each outfit. Besides a large auger to start the casing-hole, there are screw-augers, augers with ball-valves, screw-nosed cylinders with ball-valves, a T-shaped wood-cutting chisel, semicircular, cross and flat chisels for rock or bowlders, crown-borers, sand-borers and a variety of other tools, such as rod-catcher, spring-darts for tubing, etc. These tools might be much improved. The price of such an outfit at Singapore is about $685 (Mexican).

In the iron platform, arranged along the circumferences of a series of concentric circles described about the central coupling, are a number of 1-inch holes intended to permit the attachment of weights to the platform by means of hooks or lashings. When the casing has been started several feet into the ground, and a guiding-frame has been built about it, four men mount the platform, insert the boring-tools into the casing, and, walking in a circle, turn the rods by means of hand-dogs. The weight of the men and platform, plus the weights attached below, forces the casing down as the boring-tool cuts the material away. The casing, which is fastened to the platform, turns in an opposite direction from the borer by virtue of the circumstance that the men brace their feet simultaneously against the platform as they press on the hand-dogs. To guide the rods and to facilitate the lengthening and extraction of rods and tubing, a tripod, rigged with blocks and falls, is erected over the bore-hole.

The number of feet bored per hour varied according to the nature of the ground and the kind of bit. With an entirely raw crew, the average progress was from 1.33 to 2.06 feet per hour; and the time occupied in loosening, extracting, lengthening rods and tubes, etc., and necessary delays, varied from 2.27 to 11.9 times the actual boring-time. The average progress of 1.33 feet corresponds to the smaller ratio of 2.27 just given. In this case a cylindrical open shell, with screw-nosed end, was used as a bit.

When 2.06 feet average progress was made per hour, a regular screw-auger was used. This would tighten frequently, and the necessary delays amounted to nearly 12 times the actual boring-time; but the net rate of progress was greater. In both cases the soil was a tough, gray clay; the depth of holes varied from 11 to 32 feet.

The crew comprised 11 men: 4 on the platform, 6 extracting, handling rods and tubes etc., and one panning the borings.

In Banca and Billiton, where the coolies have been trained for

years by Europeans, conditions are different. In Billiton there is a regular boring corps, in charge of European engineers, which tests the fields systematically in advance, by boring first at intervals of say 100 yards, and supplementing these with holes 20 to 25 yards apart, or closer if necessary, to ascertain the course, average thickness, and character of pay-gravel. The contents of each bore-hole are carefully washed, the collected tin is weighed, and calculations are made as to the probable yield. On the basis of this estimate the fields are let out to Chinamen, against a fixed price per unit weight of tin produced * and turned over to the company.

It has been established that the average wages expended in Billiton, per *pikol* of tin-ore produced, are 20 guilders, and that 12 guilders per *pikol* is the cost of administration, etc. On the basis of these figures and the indications of the test-borings, a certain tariff is figured out, with contingent bonuses.

Under a former tariff, the basis of wages was 20 guilders per *pikol* of tin-ore, but the workman received, as minimum wages per annum, 140 guilders, or the pay for 7 *pikols*, whether he produced so much or not, 7 *pikols* being the lowest rating of annual yield per

* The unit of weight on which tin-ore is bought and sold in the East Indies is the *pikol*. The following table of native weights is compiled from the *Netherland Indies Government Almanac*, for 1890:

	Kilogrammes.
1 pikol = 100 katties,	= 61.7613
1 kattie = 16 thail,	= 0.6176
1 thail precious metal,	= 0.05409
1 koyan = 28 pikols (Samarang),	= 1729.316
1 koyan = 27 pikols (Batavia),	= 1667.555
1 koyan = 30 pikols (Soerabuja),	= 1852.839
1 koyan = 30 pikols (Batavia, for salt, etc.),	= 1852.839

The following are the figures for the Malay Peninsula, as given by E. de la Croix.

	Kilogrammes.
1 pikol = 100 katties,	= 62.500
1 kattie = 16 tahels,	= 0.625
1 tahel,	= 0.039
1 bharra = 3 pikols,	= 187.50
1 koyan = 40 pikols,	= 2500.00
16 pikols = 1 metric ton,	= 1.000

At Singapore:

1 pikol white tin metal = 60.5 kilogrammes = 133.33 lbs. av.
16.8 pikols = 2240 lbs. av. = 1 ton.

man. If he delivered the full rated quantity, or more, he received a bonus determined in a sliding scale, in addition to the regular wages of 20 guilders per *pikol*. The sliding scale was as follows:

For ground rated to yield annually per man: Pikols.	The bonus paid, besides wages, to each man delivering the rated quantity or more was: Guilders.
7	80
8	65
9	50
10	35
11	20
12	5

The reward, as will be seen, is very properly largest for the man working in the poorest ground, as workmen in richer ground, even if they do not secure the bonus, are pretty certain of getting more than the guaranteed minimum of 140 guilders.

The present minimum in Billiton is, as I understand, 120 guilders per year, representing 6-*pikol* ground; and the above sliding-scale has probably been correspondingly altered.

The pay-gravel at Kotta Ranah is sufficiently free from clay not to require puddling. It is washed in a sluice-box, 18 feet long, 3 feet wide at the head and 1 foot 5 inches wide at the tail, having a slope of 3°. Above the sluice-box a cylindrical sizer, 1 foot 6 inches in diameter and 3 feet long, made of ½-inch square bar-iron, and having a slope of 1 inch to the foot, is connected by shafting with an over-shot water-wheel. Three-quarters of the water required in sluicing is passed in with the gravel at one end of the drum; the remaining quarter drops from above, from a perforated launder, on to the revolving drum, which it enters through the ½-inch spaces, washing the coarser gravel clean. The coarse gravel is discharged at the opposite end of the drum, which has a turned-up flange, ½ inch high, to prevent a too rapid discharge of the gravel, and insure a better washing. The fine stanniferous gravel falls through the ½-inch spaces upon an apron, 2 feet 5 inches long, inclined 23°, and provided with distributors; this apron is continued for another foot, to the head of the sluice, but at 60° inclination. The gravel is raked with a flat, narrow, wooden shovel, or with forks, and is afterward skimmed. The concentrates are cleaned in the *batea*. This combination of sluice and sizer was copied from Australian mines.

The capacity of such an apparatus, when the gravel is not clayey,

was found to be 5826 pounds wet or 5194 pounds dry gravel, per hour. This average was obtained from a run of 174,786 pounds of weighed gravel.

The labor required for washing and weighing, per day, was: 1 man at ditch; 1 man wheeling away tailings; 1 man taking away coarse gravel and attending to transmission of power, etc.; 4 men washing; 2 men feeding drum on ore-floor, in all 9 men; 1 man weighing ore; 2 men carrying weighed ore to drum-floor, and half the time of 1 *mandor*. It will easily be seen that with proper arrangements, at least 4 men could be dispensed with.

Tailing samples from the sluice-box contained 0.2 pounds of black tin per ton of pay-gravel.

Singapore smelting-charges, on *long time contracts*, are indicated by the appended sliding scale.

Ore assaying	70	per cent.	and	over,	gross	price	$1.50	per	pikol,	or	$25.20	per	ton.						
"	"	65	"	"	"	"	"	"	$1.75	"	"	"	$29.40	"	"				
"	"	60	"	"	"	"	"	"	$2.00	"	"	"	$33.60	"	"				

Ores containing sulphur, copper or iron pyrites, arsenic or other impurities, requiring special treatment, must be specially arranged for. The charge for roasting is 55 cents per *pikol* extra. The deductions are, first, a small percentage for impurities, and then for loss in furnace. Deductions for trifling impurities usually vary from 0.2 to 0.5 units, and for smelting-losses the deductions are:

On Ore Assaying Per cent.	Deduction Units.
70 and upwards,	2
65 " "	3
62 " "	4
60 " "	5

For example, on a high-class ore, assaying say 74.2 per cent. of tin, there would probably be deducted 0.2 for impurities, and 2 units for loss in furnace, leaving 72 per cent. net refined tin. On ore assaying 72 per cent. and quite pure, possibly 70 per cent. net refined tin would be allowed, but more likely 69½ per cent. It will be seen that the losses thus charged are quite severe when the ore falls below a 70 per cent. grade. Assuming the deduction for impurities at 0.5 units, the losses charged really are:

For Ore Assaying Per cent.	Per cent.
70 and over,	3.57
65 " "	8.46
62 " "	10.46
60 " "	12.50

The smelters use furnaces similar to those of Cornwall. There should be a handsome profit in these charges. Their only competitors are the Chinese smelters in the local camps.

Mr. E. De la Croix gives the rates at Perak. The tin-ore there is pure, but fine, and contains no arsenic and seldom sulphur. An analysis of the Kong-Loon mineral shows tin, 66; iron, 6.15; silica, alumina, etc., 27.85. The single smelting, which suffices for the reduction of the stannic oxide to metal, is performed in a low semi-cylindrical shaft-furnace (*Spurofen*) of brick, 45 cm. in diameter and 1.85 m. high above the hearth-level, with one tuyere inclined 40° to 45°, the whole costing about $100. Smelting is done at night; 36 *pikols*, or 2.25 metric tons, of mineral are treated per night, with a consumption of 1203.125 kg. of charcoal (53.47 per cent.). The costs per *pikol* are $0.55 for labor and $2.30 for fuel, a total of $2.85. The losses in treating 66 per cent. mineral are 7.60 per cent. (much of the mineral being very fine). The slags made are ground and washed for metallic granules, or sold to special smelters. Mr. De la Croix gives the following analysis of washed and unwashed slags:

	Unwashed Slags. Per cent.	Washed Slags. Per cent.
Tin (metallic granules), ...	5.30	55
Tin silicate, ...	8.60	15

This shows that much of the silicate of tin was lost in washing.

The Chinese-built furnaces of Billiton are made of clay. I saw a similar one in Sumatra. They are short, cylindrical shaft-furnaces, 12 to 15 inches in diameter, and 3.5 to 5 feet high, from lowest hearth-level to throat. The hearth-bottom is spherically rounded, and a tuyere-hole is provided above for artificial blast.

The Chinese generally smelt during the four months of the wet season, from December to April, and during the night.

They commence preheating the furnace at about 3 P. M., charge at 7, and at 9 the charge is fused. The campaign of a set of 2 furnaces, costing when erected about 200 guilders each, is, say, 2000 *pikols*, which are run through in 80 to 90 nights. Hence the furnace-wear per day is about 5 guilders; 27½ *pikols* smelted is considered a large night's work. The cost is about 2 guilders per *pikol*, or per day, 53 guilders, divided as follows: fuel, 40; wear and tear, 5; wages, 8. The latter item includes 1 head-smelter, 4 assistants, 3 bellows-men, and 2 roustabouts, making 10 men, at average wages of 25 fl. per month.

Only about one-fourth of the Billiton tin requires refining. The black tin averages 71 per cent. in white metal, and the loss in smelting was given to me as about 3 per cent.

A word regarding the exports of tin may be added in conclusion. What is known to the trade as *Singapore tin* comes partly from Perak, partly from Sungi Ujong, Selangor, Kwalla Lumpur, Jelubu, and Malacca. Pahang is expected to add to the Singapore output hereafter. What is known as *Penang tin* is all from Perak (part of the Perak tin goes to Singapore). What is known as *Straits tin* in London is Penang and Singapore tin. What is known as *Malacca tin* in the United States is also Penang and Singapore tin, only it is *specially branded to satisfy the American buyers.* It may seem strange that Penang tin is sent to Singapore in preference to exporting it direct, since freight-rates from Penang to Singapore are 8 cents per *pikol*; insurance, $\frac{1}{8}$ per cent.; exchange and banker's commission, $\frac{1}{8}$ per cent. additional. If tin is quoted in Singapore at say $31.50 per *pikol*, the extra shipping, costs, etc., from Penang, would be 16 cents per *pikol.* Singapore overcomes this drawback of 16 cents by having better shipping-rates and more favorable banking-facilities. The shipping-rates in September, were as follows: To Holland, 15 shillings per ton; to United States, 21 shillings, to London, 7 shillings 6 pence. Insurance costs $\frac{1}{4}$ to $\frac{3}{8}$ per cent.

The exports of tin from Singapore and Penang were in *pikols* of 60.5 kg., or 133.33 lbs. avoirdupois.

1866— 95,633	1871—152,200
1867—117,745	1872—163,094
1868—102,397	1873—116,830
1869—101,889	1874—127,918
1870—102,324	

It was expected that the product of 1890 would be smaller.

According to the appendix of the government almanac for Dutch India for 1890 the tin-export from Banca and Billiton was, in kilogrammes:

	1882.	1883.	1884.	1885.	1886.	1887.	1888.
Billiton	4,520,762	3,773,263	4,144,776	4,367,570	4,100,031	5,133,458	4,992,142
Outside of Java	...	412	...	68	55	571	1,592
Banca	3,976,648	4,232,779	3,625,440	5,065,179	3,877,261	5,741,912	5,957,784

Exports of Tin from Penang and Singapore.

In pikols ($16\frac{4}{10}$ pikols = 2240 lbs).

		1883.	1884.	1885.	1886.	1887.	1888.	1889.
To United Kingdom.	Singapore..	57,180	87,075	82,700	52,377	136,264	174,234	161,674
	Penang.....	101,728	132,825	146,130	151,235	161,784	141,215	119,667
		158,908	219,900	228,830	203,612	298,048	315,449	281,341
To Continent of Europe.	Singapore..	5,762	8,971	8,850	29,783	29,233	21,691	62,303
	Penang.....	841	992	1,344	3,059	4,365	338	1,432
		6,603	9,963	10,194	32,842	33,598	22,029	63,735
To United States.	Singapore..	78,380	44,360	28,315	41,694	55,769	52,416	93,312
	Penang.....	44,998	20,592	21,394	50,896	15,990	11,106	40,289
		123,378	64,952	49,709	92,590	71,759	63,522	133,601
Total Exports.	Singapore..	141,322	140,406	119,865	123,854	221,266	248,341	317,289
	Penang.....	147,567	154,409	168,868	205,190	182,139	152,659	161,388
		288,889	294,815	288,733	329,044	403,405	401,000	478,677
		17,195 tons.	...	...	...	...	...	28,492 tons.*

* 65 per cent. increase.

The export from Billiton was in 1879, 4,412,907; in 1880, 4,604,017; and in 1881, 3,898,320 kilogrammes.

According to mercantile reports Banca exports in 1889 amounted to 5,580,000 kilos, and Billiton exports to 5,964,014 kilos.

The almanac gives the following as the amount and value of the Banca product:

	Total Produced, in 1000 Pikols.	Gross Value, in 1000 Guilders.	Net Value, in 1000 Guilders.
1882	70	5,412.5	5,121.3
1883	67	4,649.7	4,416.5
1884	67.1	4,131.1	3,922.3
1885	70.1	4,509.7	4,287.4
1886	70	5,035.5	4,785.3
1887	70	5,869.3	5,610.3
1888	70.2	5,940.9	5,686.7

According to mercantile sources the Banca and Billiton exports were as follows (in Java *pikols* of 62 kg.):

	Banca.	Billiton.
1884	65,018	70,974
1885	74,510	61,357
1886	76,552*	81,052
1887	77,090†	89,193
1888	84,397	77,840
1889	90,000	96,197

* 74.5 per cent. from Government mines.

† 70.5 per cent. from Government mines.

EXPLOSIONS FROM UNKNOWN CAUSES.

BY GEORGE ROSS GREEN, EAST ORANGE, N. J.

(Cleveland meeting, January, 1891.)

[In discussion of the paper of Mr. J. C. Bayles, read at the New York meeting of September, 1890, *Trans.*, xix., p. 18.]

IT is often so difficult to locate the causes of failures of machinery and apparatus which can be examined and studied at leisure, that it is not surprising that the causes of explosions should escape detection. Usually the explosion is the first evidence that anything is amiss, and the time for observing the conditions is necessarily brief.

An incident in a welding-works near Essen, Germany, might be cited as exemplifying what trifling causes may lead to disaster, and how difficult, under the most favorable conditions, it may be to discover them.

A blowing-engine was arranged with two suction-pipes at each end of the blowing-cylinder—one leading to the air, and the other to the gas-holder. Each pipe was provided with a valve, and all were so adjusted that the cylinder drew in gas and air in the proportions found most advantageous. The gases from the cylinder were carried directly to the furnace through suitable pipes, and while the cylinder was provided with relief-valves, every effort was made to prevent explosions, principally by compressing the gases to a point where the rate of efflux of the mixture was greater than the velocity of combustion, if the expression is here allowable.

It was found that the apparatus would work satisfactorily for some time, and then the relative proportions of the mixture would change, and an explosion might have been the result. Experiments consuming days were made to discover a method of averting this danger, but they were unsuccessful, until it was observed that the change in the composition of the mixture of gas and air took place when the gasometer began to rise or fall—the back-pressure due to friction when the holder was rising or its absence when falling being sufficient to change the conditions. As soon as this was discovered,

the remedy was soon found. A metal disc, with a comparatively small hole in its center, was placed in the gas-pipe close to the cylinder, and the gas was drawn through this hole, with so much effort that the trifling change in the gasometer-pressure was without appreciable effects.

Had an explosion occurred before this discovery was made, it would probably have been classed among the unexplainable.

A blast-pipe explosion occurred recently under very similar conditions, and in the same establishment, as that described by Dr. Bayles. I was not present at the time, but I heard the report, and was on the ground a few seconds later. The pipe is 400 feet long, of 16-inch spirally-welded tubes. Air is supplied to the pipe by a No. 9 Sturtevant pressure-blower through a Y, 130 feet from one end. The blower is in a separate room, and it draws air through holes in the roof. The pipe is tapped every 14 feet, but the last 70 feet of the longer end was not in use, and formed practically a dead end. Each end, however, of the pipe was closed with a cast-iron blank flange having a hole in its center, through which air was constantly issuing. Any gas that might find its way into the pipe would presumably escape through these holes.

The line had been in entirely successful use about a year, when, one morning in December last, the explosion took place in the dead end. The cast-iron flange was blown into many pieces, several leaving deep impressions in the brick wall, while one piece passed through a stove about 50 feet away. Fortunately no one was seriously hurt. The explosion confined itself to the last 10 feet of the dead end, and especially to the last 18 inches, which were expanded into the form of an irregular sphere—the circumference being $8\frac{1}{2}$ inches greater than formerly—but without bursting the pipe.

It was supposed that gas found its way into the pipe from a special burner that had, however, been successfully used for months; but, supposing this to be the case, why should the explosion take place in the last 18 inches of the pipe, where there was a hole for the escape of gas? Could it have been a dust-explosion, and in either case, what ignited it? No further trouble has been experienced, although the conditions remain the same, so far as known.

An explosion took place in the purifier-house of The Sherbrook Gas and Water Company's works a few months ago, under rather peculiar circumstances, and the facts in this instance may serve to illustrate other dangers of gas-purification than those in the paper under discussion.

This company has two holders; and when one was about filled, an employee was sent to turn the gas from this into the empty one. He however, closed both valves, and the gas broke the seals in the purifiers and escaped into the room. This room adjoins the generator-house, and is separated from it by a heavy brick wall, there being neither doors nor other communication between. An employee from outside started into the purifier-room; and upon his opening the door, an explosion immediately took place; the building was wrecked, and one man was killed by falling walls.

Investigation showed that there was no fire in the room; and the explanation that the explosive mixture had escaped into the generator-house through the wall and had there become ignited and communicated the flame to the gases in the purifier room was not at all satisfactory to the gas company.

The matter is still under investigation, both by the gas company and the local authorities.

This company has experienced several small explosions which they can account for in no other way, than by the theory that gas and air in certain proportions and under certain conditions are spontaneously explosive.

Explosions attended with more or less mystery occasionally occur in the sewers of cities. These explosions are due to illuminating-gas which has escaped from the mains and collected in the sewers and subways.

How it becomes ignited is usually not known—a spark from a horse's hoof or a lighted cigar has been advanced as a possible source of fire; but many of these explosions occur under conditions which necessitate our seeking other explanations.

However closely the engineer may investigate, he must acknowledge that there are many things connected with gas-explosions which are yet to be learned. During my investigation of this subject, engineers have suggested pyrophoric iron in the oxide as a cause of purifier-explosions; others suggest pyrites; and others again speak vaguely of certain subtle and unstable gaseous combinations which are readily dissociated, and cause spontaneous combustion.

While this last theory is not one that the average engineer is willing to accept, yet from the evidence presented he may draw one conclusion. It is not possible to be too careful. Many of the gas-explosions which seem so mysterious are no doubt due to quite simple causes which perhaps are overlooked because so obvious.

Another point presents itself: none but the most careful and

thoughtful men should have charge of work in gas-plants. I saw a private water-gas plant in Brooklyn a year or so ago being operated for pure water-gas. The man in charge, however, was blowing air through the coal at the same time that he was passing steam; and he was entirely unconscious that the product was a mixture of producer- and water-gases. He had erected and piped the apparatus himself (without any previous experience whatever in gas-manufacture), and had it so arranged that an explosion was both possible and probable. That he had escaped one, can be attributed only to the habit of this subtle and insidious compound to do the unexpected.

CHINESE SILVER-MINING IN MONGOLIA.

BY H. F. DAWES, ENGLEWOOD, N. J.

(Cleveland Meeting, June, 1891.)

IN China all minerals are, theoretically at least, the property of the Emperor, and the Imperial permission must be got from him for the privilege of working them. A direct tax is levied on this privilege, and by simple command, the Emperor can close a mine at any time.

Occult geomantic influences are a large factor in determining the opening or the arbitrary closing of a mine. The location with reference to points of the compass, graves and natural objects, may be really or professedly considered as indicating that those influences are favorable or unfavorable. The consequence, is, of course, that for the authorization and encouragement of any considerable enterprise, friends at court, practical politics, and more or less bribery, are required. As a result, most Chinese mining is illicit, and is carried on in a small way only. Often along a single vein, or over a small section of mining country, a few individuals obtain rights over all the openings, most probably by loaning money to the head-men of the operators and holding the mines as security for the debt. In such cases, perhaps, for better security, all the ore is reduced in the works established by the creditors, who also handle all silver produced. As the head-men are, for the most part, improvident, and the laws and customs for the collection of debts are stringent and peculiar, the "officemen" or creditors acquire in time what is practically an ownership of

the mines. Workings are often abandoned, and the title to them is passed back to the office-men. They are also transferable, with the permission of the office-men, and the new operator usually assumes all debts of the previous holder. If he is not known, he must pay the debts on taking possession; but they are a home-staying people for the most part, and unknown men seem to be the exception; so the debt usually runs on for the new man, until transferred again or cancelled.

The office-men have a large organized force, part at the reduction-works and part at the mines. The former part consists of book-keepers, store-keepers (for the office-men supply the head-men in great part with tools and materials), and representatives of the office to oversee all details of the reduction of the ore, and last, but by no means least, watchmen who are present day and night in every section or department. The mine-force usually consists of an inspector and watchman for each opening; but when the openings are isolated, or when inspectors or watchmen are increased because the mine is in *bonanza*, a cook and other functionaries are added. The inspector watches the interests of the office-men, sees that the terms of the contract are fulfilled, is on hand when ore is being taken out, reports daily on the appearance of the headings, and keeps the office thoroughly posted upon matters in his district.

The contractor or "head-man" hires his own miners, watchmen and others, as is the case in the United States.

The following is a translation of a contract between "office-men" and "head-man."

"Yang Huei Kuai asks for the old shaft of Ti Tzu Hao; he will carry out all the water and waste rock from the bottom of the shaft; he must clean it all out before beginning to work in ore.

"The first silver produced to the amount of 500 taels, no matter whether produced from ore or concentrated waste, the office-men are to receive two-and-a-half-tenths and the head-man seven-and-a-half-tenths.

"Deductions for smelting-expenses, tools and other things supplied will be made according to the old rules.

"All silver produced after the first 500 taels, no matter whether from ore or concentrated waste, the office-men are to receive three-tenths and the head-man seven-tenths.

"Deduction for smelting-expenses, tools and other things supplied will be made according to the old rules.

"If he stops work the place will be kept for him half a month, and if work is stopped for a longer time it will be given to some one else. He will not be allowed to eat the slags or shares of a new man [*i.e.*, if he forfeits his contract, he cannot approach the new man to whom the place may be given, for a sublease, or make any claim on him for debts contracted while working there]. He will not

be allowed to beat a flying hammer [*i.e.*, he must confine his work to the exact locality indicated in the contract and not work elsewhere]; he will not be allowed to keep ore in the shaft, but must send it to the office-men as fast as it is produced, and all old and new waste must be sent to the surface and not stored underground.

"It has been found in the shaft that there is a south drift, a north drift, a small north drift and others, and if they be leased to others, they will all use the same entrance. In that case he must not prevent them from using this entrance, nor in any way make trouble with them; if he does he will then be sent to the magistrate for punishment.

"This is settled face to face, publicly; and there must be no repentance.

"Work must be begun on this date.

"We keep a copy as proof, which will be sent to the head-watchman.

"Written by Hsin Chin Chung on the 3d day of the 10th moon of the 13th year of Kuang Hsu."

In general all contracts are about the same, and provide for miner-like working (though in none of them is timbering mentioned), and for forfeiture if work be suspended for a stated time. In the contract given above, there is a clause releasing any subsequent contractor from all debts contracted by the present one. The absence of this clause in many of the contracts, and its presence in this and some others, shows that some liability for old debts exists, unless disclaimed; and though such liability is described in some of the contracts to be a "bad custom," the usage still has a recognized legal weight.

In Mongolia, north of the Great Wall, there are a number of silver-mines, some run in a very small way by squatters, and others more extensively by a sort of company. The country is hilly, more or less rugged, and for the most part devoid of trees. The elevation of the narrow sandy valleys is about 1000 feet above sea-level, and the highest mountains seldom reach 4000 feet. The climate has two seasons, a wet and a dry. Apart from the almost total absence of snow in winter (a fall of two inches of light feathery snow being considered a severe snow-storm), and general dryer atmosphere, it is very like New York and vicinity in ordinary years; the summer temperature often reaching 90° F. in the shade, and during the winter being often at zero.

The miners live in one-story houses, built of stone, cemented with mud. Outside and inside are plastered with mud mixed with straw, to which lime is sometimes added; and the roofs are of thatch, not sewed on, but laid like shingles, beginning at the eaves, and weighted down with poles and stones, which do not always keep it in place against sudden gusts down the gulches. In such houses many live, huddled like sheep; but as long as they get their daily allowance of

boiled millet, with some fresh vegetables in summer-time, a few ounces of pork meat and fat twice a month, and occasional cakes of wheat flour on feast days, they seem content.

A promising outcrop is usually the starting-point of the mining work. As soon as the miners have run in on it far enough to be under cover, they have in most cases sunk on it, probably following the ore or the most promising indications. On some veins, which cut right across hills of considerable size, the conditions were good for working by adit; but in no case was advantage taken of this. The shafts are more or less irregular, though some lifts of 60 feet or more are exceptions, following small seams of spar, and in some cases merely the clay selvage. The timbers are about 3 inches in diameter, and, when not used for shoring up bad ground, are placed across the ends of the shaft about 2 feet apart and alternately from one end to the other, to serve as ladders. In unproductive ground the shafts are never more than 3 feet long. In open stopes stulls are similarly placed for climbing about on. They serve much better for ladders than the Mexican notched pole, can be travelled over more quickly and with heavier loads. When the vein is more flat, stulls are often set opposite each other, and on the foot-wall side a stick thrown across them serves as a foot-rest. When the vein is flatter yet, steps are cut directly in the foot-wall, and timbers are dispensed with entirely, unless needed for bad ground.

Cottonwood, chestnut, willow, oak, pine, and black walnut grow in the region, but not plentifully, and consequently timbers are more or less expensive. Willow seems a favorite and lasts well underground. Very few large timbers are used. In bad ground, the galleries being for the most part small, the operators seem willing to take their chances of caving; but such shoring as is used when absolutely necessary is of the flimsiest character. The galleries are about 3 feet high and 2 feet wide, just of the size that a man can easily work in, seated on the floor or a small block of wood. As little waste dirt as possible is carried to the surface, abandoned workings being usually packed with it. Even galleries in use are gradually filled, so as to leave only just room enough to get through. No drilling is done or powder used, and all work is done by hammer and chisel. The hammers are of cast-iron, weighing 6 to 8 pounds, with handles 6 inches long, and the chisels* are of iron, tipped with

* These chisels have pyramidal points and resemble the *Eisen ohne Helm* of the German miners, for which there seems to be no exact English name. *Gad* involves

steel, sharpened at the forge and tempered. The hammer-heads are worn in a short time into cup-shaped depressions, which the miners seem to think reduce the danger of missing the chisel-head. They dislike excessively to break their skins so that blood flows.

All the work is single-handed. The miner always sits while at work, with his lamp hanging from a twig wedged across the drift in a convenient position. Many of the lamps are beaten out of lead, filled with peanut oil and provided with twisted cotton wicks, which they spin themselves by hand. The waste that has to be taken to the surface is carried up in small shallow baskets on the shoulders of boys; poor ore is always taken up in this way and sorted on the surface near the mouth of the workings, while first-class ore is sacked underground and taken to the head-men in small lots as fast as extracted. Most frequently rich ore must be taken from the vein only in the presence of the inspector. When 50 or more pounds are collected it is carried to the storehouse of the office-men as provided in the contract.

Although the work is slow, the wages are low; and the head-men, by pawning their clothes and stretching their credit, can go through a good deal of unproductive ground. That which gives them the hardest fight and ultimately stops their work is water. Yet they can handle a good deal of water after their own fashion, when the lift is not too great. They place in the incline a series of boxes constructed of planks ripped out by hand and made water-tight with a putty of oil and white clay. These are set in steps about 3 feet apart, and extend from the sump to a point where the water will run out to the surface. On each of these stands a man who, with a closely-woven willow basket, holding about 4 gallons, takes the water from the box on which he stands and empties it into the one above him, and so on until it reaches the surface. Bailing goes on rapidly, with intermissions for rest, and when the work is in full swing water enough runs out to fill a 3-inch pipe. This is the hardest work in the mine, and is done by bailers, who are deeemd inferior to the miners. The stint for each bailer is 1000 baskets per shift. A considerable quantity of water can be handled in this way; but in spite of the low wages it is a serious expense, and unless the deep workings are particularly rich they are often abandoned during the rainy season, to be bailed out (if their extent does

usually the notion of a wedge, and *bit* or *drill* that of a revolving tool, which makes a hole; whereas these simply chip or dress the rock. They are from 8 to 18 inches long.

not forbid such a course) and worked again during the winter months.

The ore collected in the ore-house is taken out from time to time for treatment, the times depending on the state of the exchequer, the quantity of ore on hand and the time of the year, just before New Year being the time of the greatest activity in the smelting-yard. As a rule small lots of about 100 pounds are treated at a time. All ore is first roasted and then smelted for base bullion, after which the base bullion is refined.

The roasting-kilns are circular, built of sun-dried brick set up in open checker-work, and about 4 feet high and 6 feet in diameter. After erection they are plastered outside and inside with mud mixed with chopped straw, a place being left (to be closed with loose brick) for the removal of the charge. One such furnace lasts through many roastings.

All the brick used are of ordinary clay, rammed into wooden moulds with a heavy stone rammer, and dried in the sun. No straw is mixed with them and no water is used in their making, as the clay, when freshly dug, is moist enough to stick together under the rammer. The floor of the kilns is slightly depressed in the center, to collect the lead which is often reduced in them when rich ores are roasted. On the floor is spread a layer of charcoal a few inches thick; on this a few inches of ore, and so on until all the ore is charged. The coal is then fired and burns until burnt out. The fire is a hot one and the roasting is of necessity imperfectly done, much of the ore melting and some being reduced. The lumps in the roasted charges are broken and picked over, pieces being put aside for re-roasting, and the rest is then smelted in the blast-furnace.

The beginning of a blast-furnace is a circular depression in the clay floor of the smelting-yard, some 18 inches in diameter and 4 inches in depth, in which sifted wood ashes are pressed to make a shallow cup. Around this a wall is erected of one thickness of mud brick, set on end and plastered together with clay mixed with chopped straw. The furnaces are conical in shape, with throats about 9 inches in diameter, and stand about 3 feet high. One tuyere-hole is left near the bottom; and above this hole, on the inside of the furnace, wedge-shaped bricks are built into the wall, projecting to the center of the furnace, so that the descending charge or dropping slag may not clog the tuyere. The funnel is plastered inside and out with clay mixed with chopped straw, and when it has been

thoroughly dried and heated by a small fire in the hearth, more charcoal is put in, the wind-box is set in place and connected with the tuyere, blast is put on and charcoal and roasted ore are added, at the discretion of the head-man. Bottoms from the cupel-furnaces are also added, to furnish lead for collecting the silver; and the unreduced litharge acts also as a flux for the gangue of the ore. Slag flows from a hole near the bottom, which is kept open as long as the blast is on. Lead often runs out, sometimes from cracks that open in the stack; but the yard is clean and nothing is lost, except, of course, that which is vaporized or passes into the slag. When the lot has been run through, the wind-box is taken away, water is freely dashed in the furnace, and, when cool enough, it is torn down, the cake of lead is taken out of the bottom and the *débris* is carefully sorted over by hand. All the lead is saved, and part of the charge remaining is put aside to be crushed and washed for re-smelting. The base buillion, beaten and cleaned from adhering slag and ashes, is then ready for refining in a cupelling-furnace.

On the floor of the yard, wood-ashes are sifted through a horse-hair sieve into a conical heap some 3 feet in diameter. This is consolidated by pressure of the foot; the top is levelled off and a shallow basin is scooped in it with a wooden hoop; around the base mud-brick are set on their sides in close order, save at a place in front, which is left open to serve as a door. On top of the mound and around the basin are also set mud-brick, but in open order; and across them, so as to cover the basin, are laid mud-brick of full size; just before putting on this cover the base bullion is put in, with paper under it to keep it from bruising the ashes. Inside the little wall around the base charcoal is then piled, and enough is thrown on to give the whole a rounded dome-shape, the top being some 12 inches above the basin cover. The charcoal is ignited at the front near the door, and the whole thing is plastered over with clay mixed with straw, leaving a hole at the top 9 inches in diameter. As the bullion melts, the clay coating becomes baked by the burning charcoal, and as the cupelling continues and the charcoal settles down, the dome supports itself and confines and radiates the heat. If there be not sufficient charcoal originally to carry the refining to the end, more is added from time to time through the hole in the top. The operation is carefully watched through the doorway, and pieces of charcoal and unabsorbed dross are skimmed off with iron tools. Natural draught alone is used, and the silver produced is usually above .980 fine. Small lots of base bullion, of 10 pounds or less,

are refined on wood-ash basins, by piling charcoal about them without any covering, and using the blast from a small wind-box, the blast being delivered right over the melted bullion through a clay nozzle. The refined silver is then weighed into the office, credits are given, accounts adjusted, and it is then turned over to the melter to turn into currency.

Current silver is of two kinds, about .998 fine and about .980 fine. The finer is cast into what are called "shoes" of about 50 taels (nearly 67 oz. troy), on which are stamped the name of the makers, the name together with the external appearance being their only guaranty. Much of the silver made is not considered good enough for "shoes," so it is alloyed with copper for the lower grade and the kind most usually seen.

The melter's furnace is dome-shaped, and is built of burnt brick, open in front to the room, and having a chimney from the top running outside. It is, in principle, very like a blacksmith's forge, and is blown by a wind-box with a single tuyere. The fuel is anthracite from Peking, of pea-size, in which are imbedded the small clay cups used for melting. The cakes from the cupelling-furnaces are cut up with a cold chisel on the anvil, or, if not too thick, with large iron shears; and when the silver is fine enough, it is weighed out with the proper proportion of granulated copper into a partitioned tray for melting. The fineness is judged by the eye according to the appearance of the bullion when cut with a cold chisel, long practice having given the melters considerable skill in this particular. When not sufficiently fine, saltpeter, with some sand and salt, are added to the melted silver and the slag is skimmed off. Small ingots, about 6 ozs. in weight, are cast in hemispherical iron moulds, and these, when cleaned up, and usually deeply scored on the round side with a cold chisel, to show the interior, are current.

The only Chinese coin is the copper cash. Silver is current by weight, the unit being the tael, nominally corresponding to our ounce; but the weight of the tael varies in different parts of the country, and even between different cities in the same province; and there is also a difference between the governmental and the local tael. These discrepancies afford the means of livelihood to many money-changers or brokers.

One is often reminded of the Mexicans when watching Chinese miners. They show the same keenness in following ore, the same dislike for severe labor, and the same improvidence. The conditions under which they work, on the other hand, are different, as are also

the means for accomplishing their ends. In spite of their boasted civilization, these means seem to us extremely primitive, although long working in the same grooves has given them considerable skill, which partially offsets their obvious disadvantages.

The opening of the Tung Shang colliery, the extension of the railroad from Lu Tai to Ta Ku and Tientsin, and the building of the harbor and dry dock at Port Arthur, have drawn attention to China, and have given rise to speculation as to whether these would not prove entering wedges of western civilization, and possibly lead to the extensive working of Chinese mines after the "foreign fashion." But the Chinese are an intensely conservative people, and back of the people is the government, whose very life depends upon keeping things as they are. Hence, their policy is against granting concessions of any kind, and from the present outlook it seems extremely doubtful whether a foreigner can obtain any interest in mines of any kind.

That the country possesses great mineral wealth cannot be doubted, but until the emperor inaugurates a ceremony of pounding the drill, like that which he now performs, of turning a furrow every spring with a plow, and thereby dignifies and sanctions mining, its development promises to be desultory and slow.

METHODS OF WORKING AND SURVEYING THE MINES OF THE LONGDALE IRON COMPANY, VIRGINIA.

BY GUY R. JOHNSON, LONGDALE, VA.

(Cleveland Meeting, June, 1891.)

In view of the attention now directed to the development of the iron-ores of Virginia, and of the frequent reference in the *Transactions* of the Institute to the Longdale mines, it is presumed that a brief account of the methods of working and surveying these mines will be of interest. For a better understanding of such a description, a few words will be said concerning the geological character of the ore-deposit.

Professor Rogers says in his preliminary report of 1835, that this deposit has "more of the character of a true vein than is usually

the case with ores of this sort." According to the (now generally accepted) views of the same authority, the Oriskany sandstone (No. VII. of Rogers's classification) has here been replaced with brown hematite ore. Underlying the ore the Lower Helderberg limestone should be, and is occasionally, found as the foot-wall. Generally, however, the limestone has been replaced with a highly siliceous clay, which often runs into sandstone. Overlying the ore, and forming the hanging-wall, is the Devonian shale (No. VIII. of Rogers, the Hamilton slate of New York). This formation is encountered in great thickness, one tunnel having been driven upwards of 1200 feet through it to reach the ore. Near the ore its character changes. It becomes so much softer as to be cut readily with a knife, while the color changes to blue, red, yellow or white. Combinations of all four colors are also often met. Upon exposure to the air it swells and crumbles into fine particles, the swelling often being accompanied by the breaking-down of one or more sets of timbers in the mine.

The strike of the vein may be said to be that of the mountains, about N. 45° E., while the dip may be anything from zero to 90° in the same mine and within limits a few hundred feet apart. The general dip-direction is southeasterly. The ore crops along a distance of about one and a half miles, and has been opened in many places. Most of these workings, however, were open cuts, and have long been abandoned, the ore-supply being drawn at the present writing from two principal mines, situated, roughly speaking, at each end of the deposit, the one at the lower (S.W.) end being much the larger.

As the system of working and surveying both mines is the same, its application to the lower mine only will be described.

When mining was begun at this point, the outcrop was 500 feet above the creek-level, and was for many years worked entirely by open cuts. As time went on, however, the excavation was carried so deep (130 feet) that the cover grew too heavy and it became necessary to obtain the ore by underground workings. It was finally decided to stope the ore from the top down. This plan was adopted in 1881, and has been followed ever since with complete success.

Method of Opening the Ground.

The method of attack is to sink a test-shaft upon the ore to the depth at which it is desired to drive an adit.

The line of least distance from the surface to the ore at the level

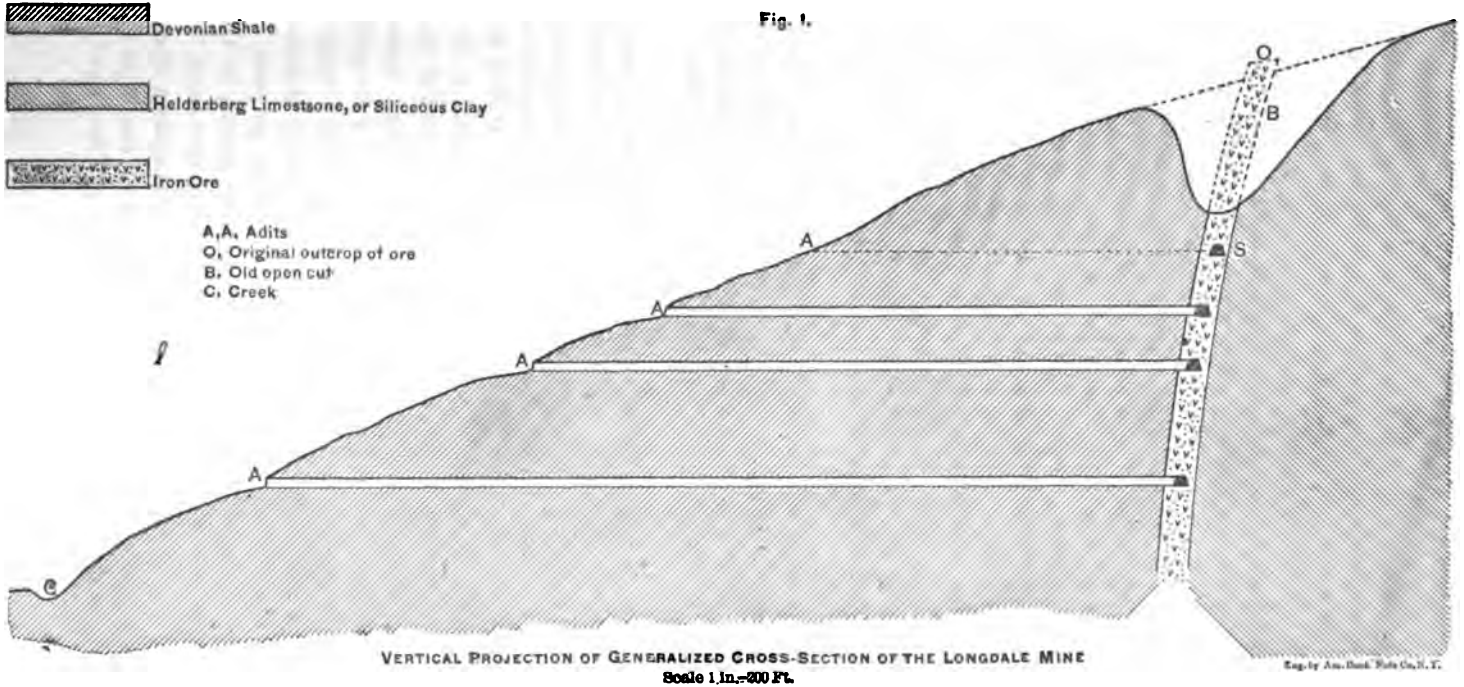
chosen (the length of which line is obviously largely governed by the height of the "lift" to be secured) is then determined, and the adit is driven straight in, through the overlying shale, to the ore, a distance varying from 300 to 1200 feet, according to the height above the creek at which the adit is driven. In Fig. 1, which illustrates the general method, the adits are represented, for the sake of clearness, as though driven in upon the same vertical plane at different levels, while in reality their entrances are from one-half mile to a mile apart, and their directions are not parallel. The highest adit was driven directly upon the ore from the bottom of a ravine which cut across the vein. Its level is indicated by a broken line in the drawing, the adit itself only showing in such a projection to the extent of its shaded cross-section at S. The shortest adit (300 feet) mentioned above is not shown in Fig. 1, as it was of secondary importance and was not used long.

On reaching the ore, the adit is continued by galleries in both directions, following the bends of the vein. These, it may be remarked, are both many and sharp; many of the sights taken during the survey of the mine being 16 to 20 feet only in length. Parallel with the main entry or car-level an air-way is driven at a height of 20 feet from the bottom of the main entry to the bottom of the air-way. As the main entries are 6 feet high, this leaves a pillar of ore 14 feet thick between the two levels. At intervals of about 120 feet on the car-level, chutes or winzes are driven up through this pillar to the air-way. These chutes are supplied with spouts and gates for loading the mine-cars. Twenty feet beyond each chute, in the direction of the heading, a second passage, called the man-way, is driven up, to afford means of ascent for the men who are working above.

As the air-way is always connected with the test-shaft above mentioned, this system affords a very perfect system of ventilation. Of course, it is necessary to keep most of the chutes and man-ways nearest the mouth of the adit closed, in order to force the incoming air to the headings. From the air-way the chutes and man-ways are driven up until they reach the surface in the bottom of the old open-cut workings. This, however, is not all done at once, but only as needed for the working of the mine.

At every 10 feet of vertical height above the air-way a lateral level 5 feet high is driven off along the ore, thus leaving a 5-foot pillar under each drift.

Fig. 1.



VERTICAL PROJECTION OF GENERALIZED CROSS-SECTION OF THE LONGDALE MINE
Scale 1 in.=200 Ft.

METHOD OF WINNING THE ORE.

When the open-cut workings, mentioned above, were about to be finally abandoned, a grillage of small poles was laid down in the manner indicated in Fig. 4. This formed a floor resting upon the ore in the bottom of the cut.

Outside working was then discontinued and the sides were suffered to fall in upon the grillage, after which the mines were ready for underground working. The system is as follows:

The pillar of ore over the highest level, or in other words, the ore left in the bottom of the old cut, is first attacked and taken out, the roof being supported during this operation by heavy props set up a short distance back of the working-face, under the grillage mentioned above, and standing upon a similar grillage placed upon the ore under foot. (Where the vein is horizontal or nearly so, the lower grillage may be omitted, the roof being allowed to fall directly upon the floor, which, in such cases, is the foot-wall). This row of posts supports the roof until the ore has been taken out for a short distance ahead of them, when a second series is set up in similar manner.

It is customary with us not to leave the roof standing, even if the props are strong enough to support it for some time, but to cause it to fall, by either "shooting out" the props (*i.e.*, breaking them down by means of dynamite) or by drawing them out if they can be saved.

The ore that comes from the face is wheeled to the nearest chute, into which it is dumped. After the top pillar has been robbed for a few feet, generally from 10 to 20, the next one below is attacked in the same way. Every level in the mine can thus be worked simultaneously, the workings resembling a series of 10-foot steps, as shown in Fig. 2, which is a vertical elevation of a part of the mine.

In this figure the shaded portion represents that part of the mine which has been stoped, the remainder being ground opened for mining but not yet stoped. The construction of a cheap floor or grillage of poles is repeated on every level to prevent the sliding of the waste from old caved stopes above. This floor is, of course, not removed after serving its temporary but important purpose.

The ore dumped into the chute descends by gravity to the car-level, unless it is stopped by meeting a flat place or "bench" in the vein and consequently in the chute. When this occurs a rehandling becomes necessary, and is carried out in one of the three following ways, as may be best suited to the ground: When the bench is

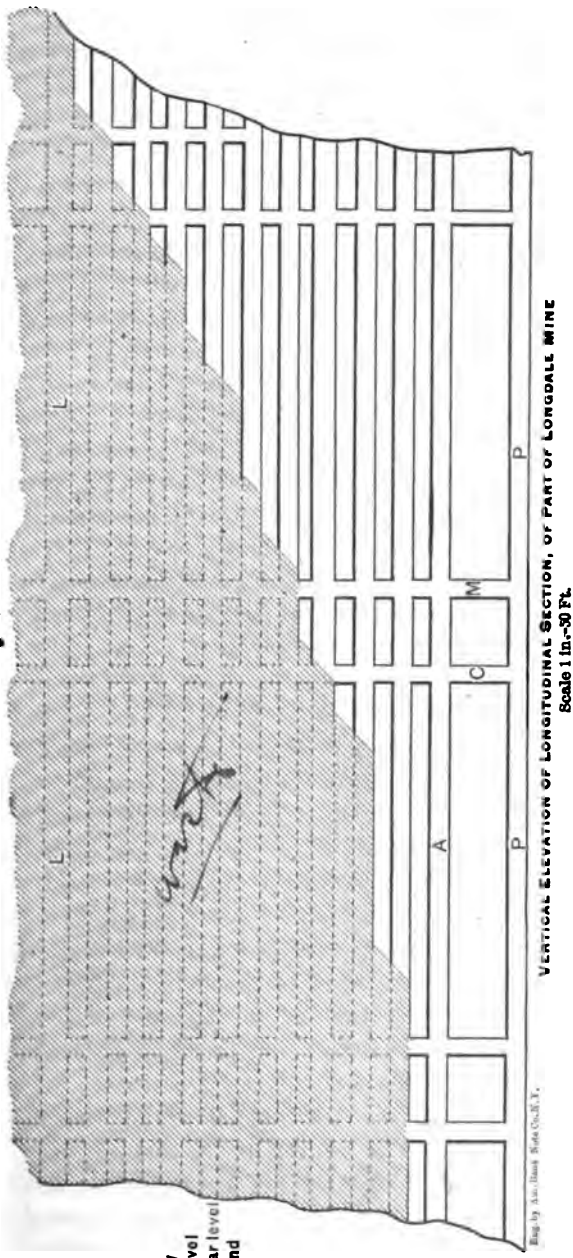


Fig. 2.

VERTICAL ELEVATION OF LONGITUDINAL SECTION, OF PART OF LONGDALE MINE
Scale 1 In.=50 Ft.

very short, one man can generally handle the ore as it comes down the chute, by shovelling it up and pitching it forward into the steep part of the chute again. If the flat place is too long for this, it is wheeled forward in wheelbarrows and again dumped into the chute. If the bench is too long to permit the economical rehandling of the ore in either of the above ways, a track is laid and a car brought into requisition. In this case the chute is provided with a spout and gate, situated at the end of the steep descent above the bench, and the ore is drawn by means of these out of the chute and into the car, which, when loaded, is pushed forward by hand to the beginning of the second steep descent in the chute and dumped.

When the ore finally arrives at the car-level, it is drawn into mine-cars, hauled to daylight and dumped upon a horizontal screen of round iron bars, which is set in the top of the ore-bin beside the railroad. The fine ore falls through the screen into a pocket, while the lumps remaining upon the bars are further broken and sorted by hand and thrown into an adjoining pocket. From these pockets the ore is drawn into cars on the railroad leading to the furnaces, the lump-ore being taken directly to the furnace-bins, while the fine is conveyed to the washer.

METHODS OF SURVEYING.

As will be seen from the foregoing description of the mining, surveying was no easy matter, for the mine was entered at four different levels, and the entrances only of these four adits could be connected by level- and transit-lines. This left the inner ends "in air." It was necessary, therefore, to survey the four car-levels with transit and level, connect their entrances by an ordinary survey and then find some way of surveying the intermediate workings, the chutes and their lateral levels; so that the inner ends of the car-levels might be thereby connected and the accuracy of the work tested. The following method, adopted after thorough discussion, has given perfectly satisfactory results:

A base-line was laid down with special care and thoroughly marked for future reference. The bearing of the line was then taken, two instruments being used, to check the accuracy of the needles. Starting from this base-line, the four entries were surveyed with the transit. The work was commenced in the lowest entry. The instrument having been set up and a back-sight taken to the next station behind the one over which the transit was placed, the telescope was revolved on its axis, the vernier-screws were loosened and the instru-

ment turned on its vernier-plate until the cross-hairs bisected the light of the plummet-lamp hanging over the station, of which the position was to be ascertained. The angle as indicated on the vernier was now read, noted and re-read, after which the bearing, as indicated by the needle, was noted. If this agreed fairly well* with the course as calculated, the assistant was sent back to the first-mentioned station, the vernier was unclamped, the telescope revolved on its axis and the instrument turned on the vernier-plates until the flame was again bisected by the cross-hairs. If the vernier then stood at zero, I was reasonably sure that no error had been made in the angle. The reason for always re-setting the instrument to zero was that it was an old-fashioned flat-centered transit, with vernier divided into four arcs of 90° each.

The angle and bearings having been taken as described, the length of the sight was measured with a steel tape. When a chute or a man-way was reached a large nail was set in the cap, or in the lagging, at the intersection of the line of the sight then being taken with the prolongation of the center-line of the upward passage. The distance to this nail from the station over which the instrument was standing was then taken, and set down in the "Notes" column, opposite the station from which the measurements were taken. This marked the lower end of the upward passage, on the line of the transit-survey; and in a similar way the entrance of the upper end into the tunnel above (or, in case it ran out into the old workings, its mouth) was noted, when the lines were run in the upper car-levels or over the outcrop in the old cuts. By this means the positions of the ends of the chutes which ran from the lower levels up to the higher ones were accurately determined.

The plummet-lamp used in this work is of the kind invented by Mr. Eckley B. Coxe, of the Institute, and described by him in his paper on "The Use of the Plummet-Lamp in Underground Surveying" (*Trans.*, i., 378.) The stations in the transit-survey were marked

* Of course, the presence of the iron rails used in the car-levels renders the needle more or less unreliable. In this connection, I would call attention to a somewhat curious experience in connection with the needle-bearings. Whenever the work was carried on in a dry section of the mine, the needle seemed but little affected by the presence of the rails; but as soon as water was encountered in sufficient quantities to make the track muddy, it became totally unreliable, variations of 5° being not uncommon. I was greatly puzzled by this, and finally became convinced that the water and mud, by making a perfect connection between the rails, permitted a greater degree of polarization and consequently increased the effect upon the needle.

with large tinned tacks, set in the ties of the track. The ties were chosen rather than the caps, because in these mines they are less apt to be disturbed by movements of the ground. Tinned tacks were employed because they were easy to find, and, being flat-headed, were not readily kicked out by the mules.

Throughout the whole survey the utmost care was used in measuring, a steel tape being employed and every precaution observed to prevent any error, either in reading or through not stretching the tape tightly enough to take out all kinks. In levelling there was nothing special to note, the most necessary precautions being to have the target always bright, have plenty of light before it, and take short sights. Whenever, in the course of levelling, an upward passage was met, the height of the top of the rail was noted, and the same thing was done at its upper end, thus fixing the elevation from top to bottom.

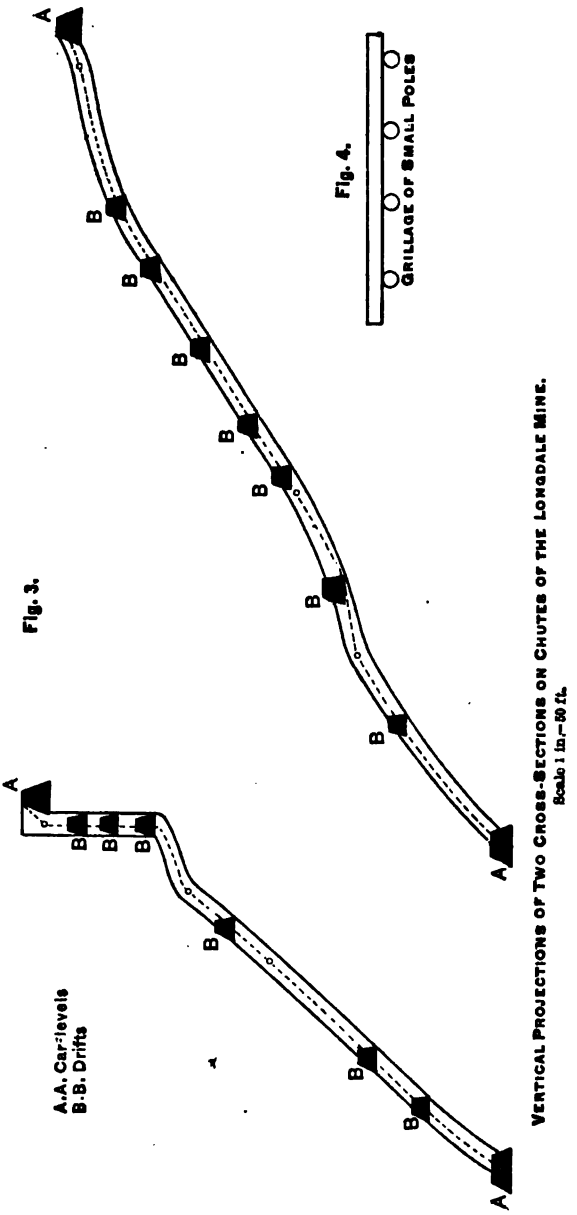
The sights averaged a little less than 3 to 100 feet. By using the above precautions, several miles of underground levelling was accomplished, and in no case was the error, as determined by "checking back" all work, greater than one-tenth of a foot.

In this way the four main levels were surveyed, after which their entrances were connected by level- and transit-lines. The chutes and their lateral levels alone remained to be surveyed, in order to enable us to make a complete map and test the accuracy of the foregoing work. To do this with transit and level was out of the question, both on account of the roughness of the chutes and also because they were much too small,—so small in places that a man could scarcely crawl through them. But the ore in this mine being brown hematite and non-magnetic, permitted the use of the magnetic needle on all levels and in all chutes where there were no rails. It was accordingly decided to use a hanging compass, in connection with a clinometer, for this portion of the survey. After several trials, I adopted the following system of using these instruments as giving the best results. Two conditions had to be met: first, the method adopted must permit accurate, yet rapid work in order to interfere with the miners as little as possible; secondly, by reason of the rough and varying character of the ground, the apparatus for suspending the compass must be both simple, light and adaptable to varying requirements. As fulfilling these conditions, the following articles were used: 100 feet of strong linen line, about one-eighth of an inch in diameter, braided to prevent much stretching or twisting, and wrapped upon a light reel, which could be easily carried. This, with

a plummet, a hatchet and nails, and three strips of wood, respectively 4, 5 and 6 feet long, constituted the equipment. The survey of a chute was preferably begun at the bottom. The distance from the top of the rail to the nail in the cap or the lagging, left when the transit-survey was made, was first noted. One end of the cord was then fastened to this rail, and the reel was carried forward, up through the chute, until the curving, either horizontally or vertically, of the chute caused the cord to diverge from the center line. One of the three wooden strips was then wedged tightly across the chute, and the cord was attached thereto and drawn very taut. A brass clip was then fastened to the line to prevent the instruments from slipping to the bottom. The compass was next suspended, the course was taken, and, the compass having been removed, the clinometer was substituted. The angle of inclination having been noted, the clinometer was taken off, and the distance from the bottom up to the strip was measured with a steel tape, always along the line. In case one or more levels were passed in the course of the sight, the distance to the center of each was noted, as was also the distance they had been driven along the ore. If they had been cut through to the next chute, then the distance from center to center between the chutes was taken, as affording another check upon the work in the final plotting. All the measures having been taken, the line was loosened at both ends, and the rear end was fastened to the strip where the forward end had been attached, while the forward end was again carried ahead and tied to another strip. At the end of the second sight, the first strip was removed and carried forward, to be used again. If the chute was timbered, the cross-sticks were nailed to the timbers instead of being wedged across. In case the chute was vertical, the plummet was used, and only the vertical measures in the chute needed to be taken. The drawings in Fig. 3 are vertical projections of two cross-sections on chutes in the mine, and give an idea of the varying slopes. The dotted lines represent the survey-lines.

Upon reaching the top of the chute, connection was made with the transit-survey by fastening, in the last sight, the forward end of the cord to the nail left when the transit-survey was made, in the line of the chute. By this means an accurate line of surveys was carried through each chute, the man-ways being located by offsets from the chutes, and it only remained to reduce the slant distances, as measured along the cord, to their vertical and horizontal values. This was

done with a trigonometer, by the use of which a vast deal of calculation was avoided.



The vertical heights between the four car-levels are, beginning at the lowest, from "Big Bend" to the "Store Tunnel," 120 feet; from the "Store Tunnel" to the "Alum Spring Tunnel," 50 feet; from the "Alum Spring Tunnel" to the top of the pillar of ore left between the bottom of the open-cut workings and the "Craft Tunnel," 56 feet; making a total of 226 feet from the lowest level to the top of the ore. I may say, in addition, that we are now driving a tunnel to take in another 120-foot lift, the ore having been tested to that depth and still continuing down.

In plotting the foregoing work, everything was reduced to terms of the magnetic meridian. The main entries were first plotted, after which the chutes were laid down. It is evident that unless the work was correct, if the chutes were plotted from the bottom up, their top ends would not fall upon the corresponding points determined in the transit-survey. This they did, with sufficient accuracy to make a reliable map of the mine.

Two projections of the mine are always made, a ground-plan or horizontal projection, and a vertical projection. The map once having been made, it is a very easy matter to keep it up to date, a week being sufficient to survey all the year's progress, both in the new work and in stoping. As the first survey and the subsequent plotting took about three months, this is another instance proving the accuracy of the proverb: "It is the first step that costs."

In conclusion, let me say for the benefit of those who have not used a hanging compass, that though the above description sounds delightfully simple, actual practice was far from being so. The character of the ground to be surveyed, which is always rough, sometimes wet, and occasionally dangerous, greatly enhanced the difficulty of doing accurate work. The keeping of the note-book in respectable shape, in fact, in any sort of shape so that it can be read afterwards, is also often almost an impossibility, owing to the streams of water which are caught by the breaking of the ground above, and pour down through the chutes, generally bringing with them a *sufficient* amount of mud. Nevertheless, if proper care be taken, this system of surveying the chutes affords very satisfactory results, in comparison with which, of course, the incidental difficulties and annoyances sink into insignificance.

A COMPOUND-PLUNGER HYDRAULIC PUMP.

BY ERNEST R. WOAKES, TOLIMA MINES, REPUBLIC OF COLUMBIA.

(Cleveland Meeting, June, 1891.)

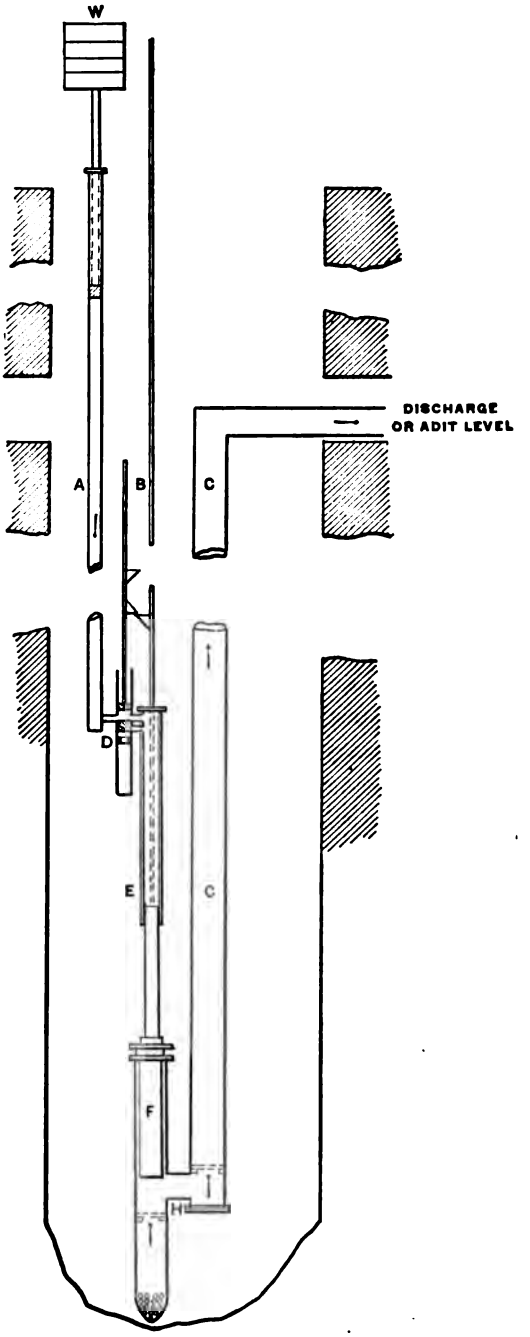
THOSE engaged in pumping from shafts, or other mining works, may be interested in the following suggestion of what is believed to be a novel method of raising moderate quantities of water against a considerable head.

It is admitted that the ordinary Cornish pump, as generally used in mines, is hard to beat, on account of both its extreme simplicity and its easy adaptation to the requirements of mining; its great drawback, however, being the enormous weight of heavy rods and successive plungers that have to be kept in motion, not to mention the necessary "balance-" and "angle-bobs," with all their accompanying friction and wear and tear.

It is contended that the following method would, to a great extent, overcome these drawbacks; though, no doubt, many old miners may say it would also introduce new ones. Possibly, this may be the case. It is only desired that this pump may be judged upon its merits.

The chief novelty of the method consists in employing the almost unlimited pressure obtainable from a high column of water, augmented, if necessary, by a hydraulic ram or press at the top, to work a small-diameter piston or plunger at the bottom of a shaft or well. This high-pressure plunger, indicated at *E*, in Fig. 1, transmits power direct to a larger-diameter plunger, *F*, constituting, together with the ordinary H-piece, valves, and wind-bore, the actual pump, which raises the water, by column *C*, to the adit-level or other discharge, placed as far as possible below the head of the high-pressure column *A* and ram *W*.

It will thus be seen, that it is only necessary to maintain a pressure on the small piston, *E*, equal to that on the larger one, *F*, by means of the small but high column of water, *A*, in order to cause *F* to descend, and thus force the water up through *C*. The up-stroke is accomplished by a small water-wheel, or other engine, connected to the light rod or wire rope *B*, this having only to raise the weight of the actual plungers and fill the H-piece with water from the tank or



COMPOUND-PLUNGER HYDRAULIC PUMP.

sump. The equilibrium-valve, shown at *D*, for regulating the admission and the cut-off of the water from the column, *A*, to the cylinder, *E*, is also worked by this rod. Of course, the amount of water used in *E* is lost at each stroke, and has to be taken into account in the amount raised by *F*. This quantity, however, is small, as will be subsequently shown. Thus it will be seen, that instead of a large engine and heavy working parts, a small one with light connections can be used; the weight of the plunger, etc., only being balanced by a small "bob" in the ordinary way. The difficulties encountered in shafts, and especially in inclines, would thus be minimized. As a set-off to this, it may be necessary, in most cases, to keep the ram running at the surface in order to maintain the pressure on the small piston, by a closed and continually descending column of water, the waste from which is maintained at the top by the ram, and which, after doing its work in the high-pressure cylinder at the bottom of the shaft, is pumped up again as far as the discharge by the large plunger which it actuates.

If desirable, two or more sets of plungers can be worked at different stations in the same shaft or well from the same high-pressure column, the sizes of various plungers being varied to suit the pressures to be overcome and the quantity of water to be raised.

In order to demonstrate more clearly the working of the pump, the following example may be taken:

A shaft 700 feet deep, with the adit- or discharge-level 100 feet below the surface, so that the water, say 100 English gallons per minute, has to be raised against a head of 600 feet.

High-pressure plunger, 3 inches diameter.

Low-pressure plunger, 10 inches diameter.

Working at 6-feet stroke, and 6 strokes per minute.

Capacity of high-pressure cylinder, 10.5 gallons per minute.

Capacity of low-pressure cylinder, 121.5 gallons per minute.

Difference, 111 gallons per minute, which is the amount of water to be raised.

The pressure on the 10-inch plunger is 260 pounds per square inch, making the total pressure on its area 20,400 pounds.

In order to produce an equal pressure on the 3-inch piston, a pressure of 2900 pounds per square inch must be maintained, which will give 20,470 pounds for the total pressure on the area of the 3-inch piston.

This pressure can be supplied by a 19 horse-power ram at the top of the column.

The power required to raise the plungers and work the valves could not be more than 5 to 6 horse-power, so that the whole system would be worked for an expenditure of 25 horse-power.

To do the same work with a Cornish pump, consisting of three plungers and one lift, would require about 30 horse-power, so that there is a slight gain in this respect with the method under discussion.

It may be said that the above pressures would be an obstacle to the successful and economical working of the system, but it is believed that by using cast-steel instead of cast-iron, in the parts subjected to high pressures, this objection would be removed. It is certain that the present tendency in all machinery is to the use of high initial pressures.

The above example could, of course, be modified to almost any extent to suit individual cases, and is only quoted to illustrate the practical application of the principle. The duplication of plungers, as before mentioned, would reduce the actual pressure on any one system.

In situations where there is a sufficient difference in altitude between the highest convenient point near the pit's mouth and the adit-or discharge-level, and where the amount of water to be raised is small, the ram at the surface could be dispensed with, the excess of natural pressure in one column overcoming that in the other.

In conclusion, it may be added that the same principle could be employed to compress air in the bottom of a shaft, the air-cylinder replacing the large plunger and casing, in which case the initial pressure required would be much less than for the pump, though, of course, the work of the mine-pump would be augmented by the amount of water used in the high-pressure cylinder.

EXPERIMENTS WITH THE IMPERATORI PROCESS AT CROTON MAGNETIC MINE, NEW YORK.

BY J. B. NAU, NEW YORK CITY.

(Cleveland Meeting, June, 1891.)

A SHORT time ago some interesting experiments concerning a new steel-making process in the open-hearth furnace were made by the writer at the Croton magnetic mine, N. Y.

The Imperatori process is a sort of mixed process in which a large amount of rich ore can be treated in presence of a metallic bath. The reasons that seem to have guided the inventor in his researches are the following: It is well known that the ordinary practice in the Martin-Siemens process consists in smelting in an open-hearth furnace, a certain amount of pig-iron together with a larger or smaller proportion of scrap. Thus, in certain works, where scrap-iron is abundant and cheap, the proportion of pig may be as low as 10 per cent., whereas, in other works, where the conditions for the supply of scrap are less favorable, the amount of scrap used in the charge may be more limited. The idea of substituting for a part of this sometimes very expensive scrap a corresponding quantity of rich iron-ore, which can be obtained at a much lower price, is quite natural. Thus the practice of the so-called pig-and-ore process consists in adding to a bath of liquid pig a certain amount of rich iron-ore, which, by its reduction, will energetically oxidize the silicon, carbon and manganese of the pig-iron.

But in the ordinary ore-process, as it is practiced, for instance, in England, the amount of ore to be added is limited by the following conditions:

1. The oxygen contained in the ore added must be only sufficient to oxidize the three elements aforesaid.

2. The amount of all the bases present (oxide of iron excepted) must be such that they will just suffice to neutralize the silicon in the pig-iron; that is to say, form a suitable slag with the silica produced by oxidizing this silicon.

These two conditions permit an easy calculation of the amount of ore required to supply the necessary oxygen for the refining of the pig-iron in the presence of a flame, which itself has a slightly oxidizing effect. Of course, the amount of ore to be added will vary according to its quality and nature. In the following calculations the magnetic ore, Fe_3O_4, is taken as a basis, this ore having been used almost exclusively in the tests to be subsequently described. It should be added that these calculations are partially based on the studies made by Professor von Ehrenwerth, of Leoben, in somewhat similar experiments carried out several years ago at Austrian iron-works.

If it be supposed, as seems to be the case in the ordinary open-hearth process, that the carbon of the pig-iron is to be entirely eliminated as CO, it follows that $1\frac{1}{3}$ parts of oxygen or 4.83 parts of magnetic

iron-ore, Fe_3O_4, would be required to oxidize one part of carbon.* But, as we have to eliminate silicon and manganese from the pig-iron besides carbon, the ore should supply also the oxygen required for the elimination of these two elements. One part of silicon requires $\frac{16}{14}$ = 1.143 of oxygen or 4.14 of Fe_3O_4, to form silica; and similarly one part of manganese will require 1.055 part of Fe_3O_4 to form MnO.

Consequently, a pig-iron containing 1.5 per cent. of silicon, 3.5 of carbon and 1.5 of manganese would require of pure magnetic oxide, Fe_3O_4:

		Per cent.
For silicon, ...	1.5×4.14	= 6.21
For carbon, ...	3.5×4.83	= 16.91
For manganese, ...	1.5×1.055	= 1.58
Total, ...		24.70

But these calculations assume that the flame in the open-hearth furnace is absolutely neutral, and that all the carbon of the pig is removed by the oxygen of the oxide of iron. Such, however, is not the case. It is a well-known fact that the flame in the open-hearth furnace is capable of oxidizing over one-half of the carbon contained in the pig-iron, so that instead of using 16.91 per cent. of Fe_3O_4 to eliminate the carbon, less than half as much would be required; and in lieu of using 24.70 of Fe_3O_4 to oxidize the aforesaid elements, 16 to 17 per cent. would be enough. On the other hand, the ore used will never be a pure magnetite, and, furthermore, all the oxide of iron contained in the ore will never be completely reduced. These considerations require a corresponding increase in the amount of ore. As a general result, in the ordinary ore-process, it has been found good practice to add about 20 per cent. of ore to the charge.

The preceding calculations are based on the amount of silicon, carbon and manganese to be removed from the pig-iron. The amounts of these elements are always below certain limits in the pig-iron; and the amount of ore added to the charge may only reach a certain maximum percentage determined by those limits. Every particle of ore added beyond would occasion a clear loss of iron. But, for well-known reasons, it would not be advisable to increase silicon and manganese in the pig beyond a certain limit. Hence

* Taking the equivalents Fe, 28 and O, 8, we have as the proportion of oxygen in Fe_3O_4, $\frac{32}{116}$ or 27.59 per cent. Hence, the number of units of Fe_3O_4 required to furnish $1\frac{1}{2}$ units of O would be $\frac{116}{32} \times \frac{3}{2} = 4.83$.

the only element that could be increased with practical advantage for the reduction of a greater amount of ore would be carbon; and as it is difficult to obtain a pig-iron or iron bath containing more than 5 per cent. of carbon, the general endeavor has been to introduce carbon with the ore. In this direction, some experiments were made in Sweden and at the iron works of Gratz in Austria, in 1881 and 1882, by adding to the pig-iron bath:

1. Briquettes formed of ore in pieces, pig-iron and coal.
2. Briquettes of pig-iron and carbon.
3. Briquettes made of carbon and finely-crushed ores agglomerated.

To impart strength and weight to these briquettes, and facilitate their sinking into the metallic bath when charged in the open-hearth furnace, it was considered indispensable that they should receive first a coating of pig-iron, and they were consequently imbedded in such a coat of pig-iron before being charged into the pig-iron bath of the steel furnace.

From these tests, Prof. von Ehrenwerth drew some very important conclusions. The ore, which had been supposed until then to have a direct decarbonizing action on the pig-iron, was found to act, on the contrary, by means of the slag it produced. The briquettes of pig-iron, ore and carbon, instead of sinking completely into the metallic bath are, on the contrary, partially dissolved in the slag, thus furnishing a highly ferruginous cinder which, by its contact with the metallic bath, deprives the latter of carbon and yields in return part of its iron. Hence a double reaction takes place between the iron bath and the ferruginous slag; on one hand a refining action on the metallic bath, and on the other a reducing action on the slag.

Although the above experiments gave partially good results, at least as far as the quality of metal was concerned, the loss of iron incurred was too heavy to permit economical working. These experiments, so far as the writer knows, had been abandoned as unsatisfactory, when, about one year ago, Mr. Imperatori, encouraged more or less by the results previously obtained, made experiments of his own.

Tests Made in America.

The Imperatori process consists substantially in treating an intimate mixture of finely pulverized rich iron-ore (at least 50 per cent. of iron) and carbonaceous materials, agglomerated into briquettes, in the presence of a metallic bath of pig-iron or carburetted iron. The relative proportions of carbon and ore are calculated in such a manner

that the carbon is present in sufficient quantities to reduce the ore directly to the metallic state, without previously being transformed into a carburetted product. The amount of carbon to be added to the ore to reduce it to metallic iron can easily be determined on the assumption that the carbon will be burnt half to CO and half to CO_2. The ores used in the tests at Croton magnetic mine were concentrates of magnetic ore, carrying, on an average, 64 per cent. of metallic iron, or 88.40 per cent. of magnetic oxide, containing 24.4 oxygen. As may easily be calculated, this amount of oxygen will burn 12.2 parts of carbon, half to CO, and half to CO_2. The coke used in the tests contained about 88 per cent. of fixed carbon, consequently, in order to obtain 12.2 parts of carbon, at least 13.86 parts of coke are required. As will be seen, a larger proportion of carbon was maintained in the experiments on account of losses, burning off at the surface, etc.

The ore had been concentrated from an ore containing only 30 per cent. of iron originally. An analysis of the concentrated ore by Messrs. Ledoux & Co., of New York, showed: Fe, 64.44; Al_2O_3, 1.01; MgO, 0.48; S, 0.88; P, 0.063; SiO_2, 6.25; Mn, not determined, but present in small quantities. Another analysis made at Croton Magnetic Mine works gave: Fe, 65.00; P, 0.064; S, 1.136. It may be said at once that the sulphur in these ores is extremely high, ranging between 0.88 and 1.136, an average of 1 per cent., much too large a proportion to give good results in the open-hearth process, at least as far as physical tests are concerned.

The high percentage of sulphur in the ores used was, in fact, more than double the amount that had been expected, 0.50 per cent. having been considered and stated as the maximum. This was an accidental occurrence, mainly due to an imperfect and too short roasting. The ores roasted since, and containing mostly much less than 0.50 per cent. sulphur, were not available at the time the tests were made.

Coal and coke used in the briquettes were of good quality; the coal contained only 4.26 of ash, and 0.643 S. The ash in the coke varied between 9.45 and 10.15, and the sulphur between 0.59 and 0.85, while phosphorus was very low.

Manufacture of the Briquettes.—Ore and carbonaceous materials were first crushed to powder and then intimately mixed by hand, and in the following proportions by weight:

(1) 100 parts of ore to 25 parts of coke.
(2) " " 30 " coal.
(3) " " 21 " coke.
(4) " " 27 " coal.

In order to obtain these proportions easily, and without losing time in accurate weighing, boxes were built which, when filled with ore and coke or coal respectively, gave the desired proportion in each case.

Lime, to the extent of a little over 1 per cent. of the weight of ore, was stirred in water, and then added to the mixture, in order to constitute a flux of a given nature. The mass, after stirring, was formed by hand in special moulds into briquettes of 50 to 60 pounds each. These were left to dry in the air for 10 days at least, under cover; and about 24 hours before being used in the furnace, they were put in a drying-oven, in order to remove the last traces of water and to heat them to such a point that they could still be charged by hand in the furnace.

The slow drying of the briquettes imparted to them a certain compactness due to the more or less complete hydration of the iron-ore. Furthermore, it was intended to render less objectionable the presence of the large amount of sulphur by transforming, partly through prolonged exposure to the air in presence of humidity, the sulphide of iron in the ore-mixture into sulphate of iron, in which state it can be eliminated with the slag during the heat. All the heats made at Croton magnetic mine were, in their general features, carried on in the following manner:

Conduct of the Heats.—First, the whole amount of pig-iron and scrap to be used in the heat was charged at once and allowed to heat up. When it began to become pasty and to melt partly, the charging of the ore-briquettes still warm from the drying-oven was begun. These briquettes were laid side by side, as evenly as possible, over the surface of the metal bath into which they at once partially sank. When, for any reason, the briquettes could not be charged all at once, the heat was left to itself for about half an hour, and then the remaining briquettes were charged in places where they could sink at once partially into the bath.

As soon as the whole charge was in the furnace, the heat was left to itself. In most of the heats, but specially in the later ones, the reaction began at once. Numerous blue flames covered the whole surface. The cinder, which at the beginning of the heat was black and very rich in iron, brightened up towards the end of the heat and appeared either dark bottle-green or light yellowish-green, like blast-furnace cinder.

In order to obtain a quick reaction between the carbon of the metallic bath and the layer of the rich ferruginous cinder on top of

it, the maintenance of a very high temperature in the furnace is essential. Otherwise the reaction will be slow, and the reduction of the iron from the ferruginous cinder incomplete. On the other hand, during an unduly prolonged heat the acid lining of the furnace will be destroyed very rapidly. Tests of metal were taken from time to time, and ore in lumps was added whenever it was found necessary. In the last heats made, the amount of ore thus added in lumps was largely reduced, and served rather for refining purposes. As soon as the tests of metal and the quietness of the bath in the furnace indicated a sufficient decarburization, ferro-manganese was added, and the metal was cast as in the ordinary open-hearth process.

It should be said in a general way that the whole series of ten heats was made under very unfavorable conditions. The works at Croton mine contain one 10-ton open-hearth furnace, which had been standing idle for about a year. The bottom, made of siliceous sand, was still full of metal and cinder from previous tests, made in the furnace by another process, and though it had been prepared as far as practicable it was still far from perfect during the first three heats.

Moreover, on account of the limited number of test-heats to be made, and in order to reduce the number of men employed, it had been decided to work only in the daytime, and consequently to make only one heat in 24 hours. But it was found for several reasons impracticable to make during the first week more than one heat every 48 hours; and thus the furnace was allowed to cool off, to the great detriment of the following heat. It must also be mentioned that during the week of the first tests, beginning March 9th, as well as during the preceding weeks, we had very heavy frosts, and as no special arrangement had been provided for making the briquettes in a place where they could be kept from freezing, they were prevented from drying slowly and for a certain length of time, as required. Indeed, it was actually necessary to thaw them out and dry them in a drying-oven for 24 to 36 hours previous to their being charged into the furnace. The consequence was that they crumbled partly to pieces during the charging, and instead of sinking partially in the bath of pig-iron and being surrounded on all sides by the heat and flame, the mixture of ore and carbonaceous materials covered the metal with a thick layer, conducting heat badly. If to this drawback we add the fact previously mentioned that, on account of the long intervals between the heats made during the first week, it was difficult to obtain a high temperature in the furnace, the higher

losses obtained during this first week can easily be explained. This loss, indeed, amounted to nearly 24 per cent. in the first heat, of which the duration was 9 hours 45 minutes, and 20.1 per cent. in the second heat, lasting 9 hours 55 minutes. In the third heat, on account of a chilled steel-ladle, the metal could not be cast entirely, and the loss could not be accurately determined. This heat lasted 13 hours 30 minutes.

The three following tests, Nos. 4, 5 and 6, were made on three consecutive days. The furnace was in better shape and had been more thoroughly heated; the bottom had been entirely remade, and the ore-briquettes had been dried more slowly and were warm when charged. Some of them still showed the effects of previous freezing, but comparatively few broke partially when falling in the bath. They were charged after the smelting of the pig-iron and scrap had begun, and consequently sank at once somewhat in the bath. The results of these heats were much more satisfactory, except for charge No. 4, the first made on the new bottom. In this heat the furnace was not thoroughly hot. The heat lasted 11 hours 25 minutes, and the total loss amounted to 25 per cent.

Details of Charges.

Charge No. 4, March 16, 1891.

	Total charge, lbs.		Iron contained, lbs.
Port Henry pig-iron,	4000		4000.00
Coal- and ore-briquettes (100 ore to 30 coal)	3400	(64 per cent. iron in ore)	1673.60
Scrap-iron,	2000	(deduction of 6 per cent.)	1880.00
Mokta-ore,	1200	(55 per cent. iron)	660.00
Spiegel,	100		100.00
Ferro-Manganese,	160		160.00
Lime and Limestone,	80		
Total iron in charge,			8473.60
Product obtained: Good ingots,	5505		
Scrap,	846		
			6351.00
Total loss (25 per cent.),			2122.60

Total amount of ore in the charge,	3815 pounds.
Proportion of ore to the total charge,	37.80 per cent.
Total time of charge,	11 hours, 25 minutes
Yield of iron from ore, about	20 per cent.

Analysis of steel: C, 0.18; P, 0.148; Mn, 1.141; S, 0.229.

Charge No. 5, *March* 17, 1891.

	Total charge, lbs.	Iron contained, lbs.
Port Henry pig-iron,	2500	2500
Scrap-iron,	3500 (6 per cent. deducted),	3290
Coke- and ore-briquettes (100 ore to 25 coke),	3750 (ore at 64 per cent. iron),	1920
Spiegel,	100	100
Ferro-Manganese,	175	175
Lime,	50	
Total iron in charge,		7985
Product obtained: Good ingots,	6073	
Scrap,	664	
		6737
Total loss (15.60 per cent.),		1248

Total amount of ore in charge, ... 3000 pounds.
Proportion of ore to the total charge, ... 32.30 per cent.
Total time of charge, ... 7 hours (from charging to tapping).
Yield of iron from the ore, nearly . . . . . 40 per cent.
Analysis of steel: C, 0.20; P, 0.161; Mn, 0.952; S, 0.314.

Charge No. 6, *March* 18, 1891.

	Total charge. lbs.	Reduction to iron. lbs.
Port Henry pig,	3000	3000
Old ingots,	1734	1734
Scrap,	1266	1190
Ore- and coke-briquettes (100 ore to 25 coke),	3350	1715
Mokta-ore (55 per cent. iron),	250	138
Ferro-Manganese,	155	155
Lime,	50	
Total iron,		7932
Total steel obtained: Good ingots,	6643	
Scrap,	563	
		7206
Total loss (9.16 per cent.),		726

Amount of ore in the charge, ... 2930 pounds.
Proportion of ore to the total charge, ... 32.2 per cent.
Total time of charge, ... 7 hours, 50 minutes.
Yield of iron from the ore, ... 53.8 per cent.
Analysis of steel: C, 0.25; P, 0.152; Mn, 0.890; S, 0.342.

These three heats enable us already to draw some conclusions as to the value of the process.

During the heat No. 4, as already observed, the furnace was not yet heated thoroughly, hence a very heavy loss, 25 per cent., and a

very long heat, during all of which, but especially for the first 6 or 7 hours, the cinder was thick and pasty, and although most of the iron was reduced from the ore, a large amount of it was found afterwards in the slag in the shape of round, metallic globules, surrounded by the thick cinder, and thus prevented from reaching the iron bath. In fact, the cinder was too cold.

During the 5th heat the furnace was much hotter, the reaction between the slag and metal was livelier, and less iron (though still too much) was found in globules in the slag. The duration of the heat was considerably shortened, and the loss reduced to 15.6 per cent. But it was even as low as 9.16 per cent. in No. 6, when the furnace was thoroughly heated up. The reactions in this heat were very lively, the cinder perfectly fluid, less iron carried in globules. This charge lasted 7 hours, 50 minutes, and would have shown a loss still smaller but for the fact that towards the end the bottom rose partly, and interfered with the reaction.

In heats Nos. 1, 2, 3 and 4, the cinder had a heavy, dark, glassy look, and contained, in some cases, as much as 35 per cent. of metallic iron. Besides, the cinder directly in contact with the metallic bath contained even more iron, and enclosed at the same time much reduced iron in globules. The cinder in heats Nos. 5 and 6, on the contrary, was glassy and light yellowish-brown. That of No. 5 was analyzed by Messrs. Ledoux & Co., of New York, as follows:

SiO_2, 57.21		
Al_2O_3, 6.26		
FeO, 28.77	Fe = 22.38	Sesqui-acid silicate.
MnO, 2.95	Mn = 2.30	
CaO, 3.08		
MgO, 0.97		
99.24		

This cinder carried, besides, 9.50 per cent. of metallic iron, distributed in globules through the entire mass.

The cinder from No. 6 has not been analyzed, but in composition and appearance can safely be compared to that of No. 5. However, this charge having been still hotter than No. 5, the slag was more liquid, and carried considerably less iron in globules. This partly explains the smaller loss observed in this heat.

After heat No. 6, work had to be stopped for five days on account of lack of gas-coal and suitable sand for the bottoms. During this stoppage the bottom was almost completely renewed, and no iron from previous heats was left in it. But the sand received and used

for this work was not of good refractory quality, and caused the bottom to partially rise during the following heat. The furnace was thoroughly heated up, the ore- and coke-briquettes used in this charge, having suffered very little from the frost, had been allowed to dry slowly, and were taken directly, and as warm as possible, from the drying-oven and charged into the partly molten bath. The reaction between pig-iron bath and ore took place at once and in a very lively manner. The details of this heat are given below:

Charge No. 7, March 23, 1891.

	Total charge, lbs.	Iron contained, lbs.
Port Henry pig-iron,	3000	3000.00
Old ingots and steel scrap,	3000	3000.00
Coke and ore-briquettes (100 ore, 25 coke),	3850 (64 per cent. iron)	1961.00
Mokta ore,	385 (55 per cent. iron)	211.75
Ferro-manganese,	240	240.00
Lime,	30	
Total iron in charge,		8412.75
Total product: Good ingots,	6150	
Scrap,	1400	
		7550.00
Total loss (10.2 per cent),		862.00

Amount of ore in the charge,	3465 pounds.
Proportion of ore to the total charge,	35.6 per cent.
Total time of charge,	9 hours.
Yield of iron from the ore,	52.17 per cent.

Analysis of steel: C, 0.20; P, 0.084; Mn, 1.727; S, 0.286.

The manganese in this steel is rather high. This was done intentionally. While this heat was in progress we received the first analysis, which showed that sulphur, instead of being lower than 0.50 per cent. in the ore, as had been claimed, amounted really to about 1 per cent. The steel in charge No. 6 had been found to be red-short, on account of the high percentage of sulphur. Therefore, to correct as much as possible the bad effects of sulphur in charge No. 7, the percentage of ferro-manganese added was increased.

During all this heat, the temperature in the furnace was very high; and had it not been for the upheaving of the bottom towards the end of the heat, on account of the bad sand, the duration would not have exceeded about 8 hours. Moreover, a part of the bottom covering the metallic bath towards the end, the reaction became somewhat slower.

Nevertheless, the results of this heat can be considered as very important. The bottom having been entirely renewed, could not contain any iron from previous charges; hence all the metal obtained had really been yielded solely by the iron and ore charged. Notwithstanding the trouble occasioned by the uprising of the bottom, the loss was only 10.2 per cent. Of the 64 per cent. of iron contained in the ore, 52.17 per cent. were obtained.

The cinder of this charge had a light yellow-greenish color, and contained less iron than that of No. 5. Manganese, as indicated by the greenish tint of the slag, was higher than that of No. 5. The cinder still contained 7 to 8 per cent. of metallic iron in globules. These globules of iron would have sunk into the metallic bath had the cinder been thin and liquid enough; and the loss would have been considerably reduced thereby.

Analysis of the cinder showed:

SiO_2,	59.93		
Al_2O_3,	1.55		
CaO,	2.75		
MnO,	5.37	Mn = 4.16	Sesqui-acid silicate.
FeO,	28.01	Fe = 21.78	
MgO,	small quantity.		

After this heat, work had to be stopped again for lack of sand. It was resumed a week later, when an ordinary pig and scrap Martin-Siemens heat was made, no briquette or ore being used. This heat, the first one made in the furnace after a stoppage of a week, showed a loss of 15.5 per cent. The two last heats, Nos. 9 and 10, were made two weeks later. These two heats were interesting, in that a new mixture of ore and carbon was tried. In heat No. 9, ore- and coke briquettes were used, containing 21 parts of coke to 100 parts of ore. In heat No. 10, ore- and coal-briquettes were used, containing 27 parts of coal to 100 ore. The briquettes of both these heats had been slowly and completely dried in the open air first, and then for about 24 hours in the drying-oven. They had a dull metallic ring and were hard and resistant. The sand used for the bottom was the same as had been used in heat No. 7. In these two heats efforts were made to eliminate in the open-hearth furnace part of the sulphur introduced by the ore; sulphur which had influenced the quality of the steel obtained in the previous heats. It is a well-known fact that when two pig-irons, the one containing sulphur, the other containing manganese, are smelted together, both these ele-

ments will be eliminated simultaneously in the slag as sulphide of manganese.

In these tests we had a pig-iron very free from sulphur and an ore, in the shape of briquettes, containing an average of 1 per cent. of sulphur. It had been found by calculation that the metallic bath obtained after the pig-iron and the ore-briquettes had been smelted would contain about 0.50 per cent. of sulphur, and that in heat No. 9, for instance, it would require about 69 pounds of manganese besides that already contained in the original pig to remove the sulphur as sulphide of manganese. The manganese was added in the shape of ferro-manganese at the beginning of the charge. The same was done in charge No. 10. At the same time, and partially for the same reasons, some fluor-spar was added, which was expected also to render the cinder more liquid.

It must be confessed that the action of manganese on the sulphur in both these charges did not seem to be very effective. This may probably be attributed to the fact that the manganese in the melted bath had time enough to combine with silica before it came in contact with the sulphur in the reducing ore, and was already in the slag before the ore was entirely reduced.

Charge No. 9, *April* 16, 1891.

	Total charge, lbs.	Iron contained, lbs.
Pig-iron,	3000	3000
Old ingots,	1836	1836
Steel scrap,	1664	1664
Coke- and ore-briquettes (21 coke, 100 ore),	4100 (ore 64 per cent.)	2168
Ferro-manganese,	230	230
Mokta ore,	600 (55 per cent. iron)	330
Fluor-spar,	36	
Lime,	40	
Total iron in charge,		9228
Total steel: Good ingots,	6278	
Scrap,	487	
		6765
Total loss (26.6 per cent.),		2463

Amount of ore in charge, ... 3988 pounds.
Proportion of ore to the total charge, ... 37.2 per cent.
Total time of charge, ... 8 hours, 20 minutes.
Yield of iron from the ore, ... 13.8 per cent.

Analysis of steel: C, 0.26; P, 0.099; Mn, 0.848; S, 0.389.

This charge was made on an entirely new bottom, which, when the charging had been completed, did not seem to be entirely heated through. Nevertheless, the reaction between the ore and pig was very hot and active. However, after about 6 hours from the beginning of the charging, the bottom upheaved and interfered completely with the heat. The reaction during the last two hours became slow and dull on account of part of the bottom covering the bath and preventing the heat from penetrating and reaching the bath. The slag became thicker and held in its mass much of the reduced iron in the shape of globules which was carried off and contributed largely to increase the loss.

Charge No. 10, *April* 17, 1891.

This charge being the last one was largely increased in weight for different reasons. The furnace was very hot and in good shape to carry a heavier charge. Moreover, it was expected that the lining of the furnace would be just as likely to be destroyed by a small charge as by a heavier one, and that in such a case, a heavier charge would give proportionally better results and less loss.

Details of the Charge.

	Total charge, lbs.	Iron contained, lbs.
Port Henry pig,	5000	5000.00
Small wrought-iron scrap,	5000 (less 10 per cent. loss)	4500.00
Coke- and ore-briquettes,	5100	2570.00
Mokta ore,	450	247.50
Ferro-manganese,	305	305.00
Fluor-spar,	40	
Lime,	40	
Total iron in charge,		12,622.50
Total product: Good ingots,	10,935	
Scrap,	1069	12,004.00
Total loss (4.90 per cent.),		618.50

Amount of ore in the charge,	4466 pounds.
Proportion of ore to the charge,	30 per cent.
Total time of charge,	9 hours 50 minutes.
Yield of iron from the ore,	66.6 per cent.

Analysis of steel; C, 0.310; P, 0.166; Mn, 0.523; S, 0.256.

In this charge the pig-iron and scrap were almost completely melted when the charging of the briquettes was begun. The first briquettes charged sank immediately in the iron bath. A strong

reaction took place at once between the pig-iron and the briquettes, and the latter could be seen slowly whirling all over the bath, surrounded by long blue flames. The level of the bath rose nearly to the height of the charging-doors, at such a rate that some of the briquettes were nearly completely covered by the rising cinder and could best be located by the numerous long blue flames emerging from the bath at each particular spot so occupied. All the briquettes, when charged, had been taken directly from the drying-oven and were heated throughout. The slag in this charge, as well as in No. 9, was very thin and liquid. Six hours after charge No. 10 was put in the furnace, and when the boiling had already begun to subside, the new sand put on the bottom before the charge came again to the surface and covered the metal, stopping almost completely all the reaction. Had this not happened the charge would have been entirely finished after 7½ to 8 hours.

The cinder of this last charge, on account of its great fluidity, contained very little reduced iron in globules. Besides, the temperature throughout the heat having been very high, some of the iron that had settled during the previous charges on the bottom became liquid and thus increased the weight of the final product. This explains the very light loss obtained in this charge, and also why it has been found that an ore known to have contained only 64 per cent. of iron apparently yielded 66.6 per cent.

Conclusions.

These tests allow us to draw the following conclusions:

The last six heats mentioned above gave losses varying from 26.6 per cent. the heaviest loss observed, down to 4.90 per cent.; in other words, the general loss would be equal to 14.07 per cent. We also notice that these losses are heaviest in the first charges made on a new bottom or after the furnace has been standing idle for a few days; in fact, they are heaviest whenever the furnace is not hot enough throughout. This teaches us that, in order to reduce the losses, a very high temperature has to be maintained, allowing the reactions between iron and slag to be rapid.

Furthermore, the two slags analyzed completely, as well as the observations made on the slags not analyzed, invariably showed metallic iron scattered throughout the slag in the shape of iron globules. In the two slags analyzed we found that the metallic iron constituted from 7 to 9.50 per cent. of the weight of the cinder. This can easily be avoided by heating the furnace thoroughly and

keeping the temperature very high throughout the operation. At the same time a judicious addition of lime and fluor-spar will impart greater thinness and fluidity to the slag, thus allowing the reduced iron to sink at once and be absorbed by the metallic bath.

In No. 10 the furnace was very hot throughout the heat, and the cinder, on account of the presence of fluor spar, was very liquid. For these reasons very little iron was carried out in the shape of globules.

In heats 7, 9 and 10 great trouble was always experienced from the upheaving of the bottom. This can only be attributed to the bad sand we had received before heat No. 7. This sand was not as refractory as the sand used in the preceding charges.

Heat No. 7, which was made on an entirely new bottom containing no iron from previous charges, gave a loss of only 10.2 per cent., and yet some iron globules were carried off with the cinder. Hence this charge proves that in good practice and in continuous working, when the furnace is constantly kept at a high temperature, a loss of less than 10 per cent. can be depended upon; especially when, at the same time, some means are taken to obtain a fluid cinder.

When, as already stated, we consider that we had to work in very unfavorable circumstances, in an open-hearth furnace which had been standing idle for a long time; that the different charges on account of a very irregular and most defective supply of raw materials, had to be made at long intervals; that the briquettes used in first 7 charges had greatly suffered from the frost; that the sand used in the last heats, not being refractory enough, gave us great trouble, and that most of the men were new hands and only temporarily engaged for the special purpose of the tests, then we may say that an average loss of only 14.7 per cent. during these tests is indeed very satisfactory.*

In a pamphlet published by Mr. C. Helson, and commented on by Mr. Howe in the second edition of his *Metallurgy of Steel*, 1891, vol. i., p. 384, the results of some of the heats made with this process in Europe gave losses as follows:

	Per cent.
First heat,	13 30
Second heat,	7.00
Third heat,	6.40

* This average has been obtained, not by taking the mean of the various tests, but by taking the total iron charged and the total iron produced in all of them.

An average of only 9 per cent.

Should a neutral instead of a basic lining be used, then the composition of the cinder could be calculated beforehand in such a manner that the place of iron in the cinder could be taken up by earthy bases such as lime, magnesia, etc., and not only could the yield in iron be increased, but the amount of cinder considerably diminished.

Whenever a slag rich in oxide of iron, as was the case in these tests, comes in contact with a siliceous lining at a high temperature, it will be impossible to obtain a cinder of a basic or neutral character. Thus the slag that we intended to obtain in the tests was a singulo-silicate, whereas the real slags as shown by two analyses, were sesqui-silicates.

With a neutral lining nothing of this kind could happen; and consequently the loss of iron in a judiciously-conducted heat, made under good conditions, would always be less than 10 per cent.

The use of such a lining would not only be beneficial on account of the reduced loss in iron, but also on account of its power to resist the action of the cinder. It would require but little repairs and thus shorten the duration of the heat. Such a lining, made of chromic iron, has been in use for some years in Europe in many steel-works, where it gives the best results.

But even on an acid lining the duration of the heat which, in these tests, varied from 7 hours to 11 hours, 25 minutes, averaging 8 hours, 50 minutes, could be kept within reasonable limits if, with a good quality of sand and a lining made with proper care, the following precautions were taken:

1. The briquettes should be slowly and completely dried by exposure to the air under cover from 8 to 10 days. Of course, in current practice, the briquettes, instead of being made by hand as was the case in the above tests, would be made with special machinery, which method, besides the advantage of a greater economy, would secure stronger and more resistant products.

2. The dried briquettes, for 24 hours previous to their being used in the furnace, should be thoroughly heated in a drying-oven, but only to such a degree that they could be still charged by hand.

3. The briquettes should be charged only after the pig-iron and scrap previously charged on the bottom of the furnace have already constituted by their partial fusion a metallic bath into which they can be dumped and will partially sink, then giving rise to an immediate action between the carbon and the ore and allowing the

formation of iron sponge, smelted at once and as fast as produced, by the surrounding bath, without danger of reoxidation.

4. Finally, the furnace should be thoroughly heated before charging is begun, and the temperature must be maintained as high as possible while the heat is progressing.

Quality of Steel Obtained.

The only ores that we had at our disposal contained, on an average, 1 per cent. of sulphur, a proportion much too high for good results. The wrought- and steel-scrap used contained much sulphur and phosphorus—elements that cannot be removed in the acid open-hearth process. It is sufficiently well known that, by suitable preliminary choice of treatment, material low in phosphorus and sulphur can be secured.

The only element, therefore, the elimination of which is interesting to follow in these tests, is carbon.

The following table gives the percentages of carbon in the steel obtained in the 10 heats made:

	Per cent. carbon.
No. 1,	0.69
" 2,	0.15
" 3,	0.19
" 4,	0.18
" 5,	0.20
" 6,	0.25
" 7,	0.20
" 8,	0.28
" 9,	0.26
" 10,	0.31

The first heat was stopped intentionally before the steel was soft, it being already late in the evening. In all the other heats carbon varied from 0.15 to 0.31 per cent.

This result was obtained without the least difficulty. It is, indeed, as easy as in the ordinary open-hearth process to bring the carbon to within certain limits. As soon as the ore is smelted the operation is conducted exactly as for an ordinary open-hearth heat, and the layer of cinder on top of the metal is, in fact, no thicker than it would be in the basic open-hearth.

In some of the heats manganese is very high. This was done, as has been explained, intentionally, in order to counteract partly the bad effect of the excess of sulphur in the ores used.

Physical tests of the steel were not made; but, on account of the occasional large percentage of sulphur in the ores during the tests (a percentage much above what these concentrates contain ordinarily), such tests would probably not have been satisfactory.

The proportion of ore in the charges varied from 30 to 46.70 and was, on an average, 36.7 per cent.

As already remarked, four different mixtures of ore and carbonaceous materials were tried. The briquettes in the last two charges contained substantially 21 parts of coke and 27 parts of coal, respectively, to 100 parts of ore. These mixtures gave good results.

Comparisons of Cost between the Imperatori and the Ordinary Martin-Siemens Process.

The cost of one ton of steel in the two processes depends on:

1. The prices of the raw materials used in the charge and their relative proportion.
2. Labor.
3. Furnace-repairs.
4. The number of charges made in twenty-four hours.

As to the price and proportions of raw materials, it has been found at the steel-works of Savona, Italy, where the Imperatori process was first used, and where the excellent results obtained since its introduction, both as to quality of steel and economy of operation, have led to its final adoption in current daily practice, that the amount of scrap used in the charge ought not to be higher than about 25 per cent., and even less if possible. The average composition of a charge in this process may be given as follows: 40 pig-iron; 20 scrap; 40 ore in briquettes.

In the ordinary open-hearth furnace the composition of the charge varies largely with the supply of scrap, but may be taken to be, on an average, 30 pig-iron to 70 scrap.

We may assume for a calculation the price of pig-iron at $19 a ton, and that of scrap-iron at $21 a ton. The cost of the ore-briquettes may be estimated as follows:

Ore, crushed, ground and concentrated to 64 per cent. of iron can easily be obtained in ordinary practice at 5 cts. a unit, and in many places at a cheaper rate,* so that a ton of such ore will cost $3.20. Coke can be obtained at the rate of $4 a ton and anthracite coal,

* See "Progress in Magnetic Concentration of Iron-Ore," by John Birkinbine.—*Trans.*, xix., 656.

which would be very good to use in these briquettes, is as cheap or cheaper. But as the coke or coal has to be ground to a fine powder, we will suppose it to cost even $5 a ton. Briquettes of a good composition will contain about 21 parts coke to 100 ore; in other words, a ton of ore will require 0.21 ton of coke. Briquettes containing a ton of ore will then cost:

1 ton of ore,	$3.20
0.21 ton of coke @ $5,	1.05
	$4.25

But a ton of metallic iron requires the use of 1.56 tons of 64 per cent. ore, and will cost $1.56 \times \$4.25 = \$6.63$.

The labor in both cases will be the same, except for the making and drying of the briquettes. In current practice, these briquettes will be made by machinery, and it will require only a few men to make from 800 to 1000 briquettes a day, weighing each 50 to 60 pounds, so that they can be made very cheap. As a liberal estimate, we will take 50 cts. per ton of ore, so that the final expense per ton of iron in the ore would amount to:

Ore and coal (both ground),	$6.63
Labor on 1.56 ton of ore at 50 cts. per ton,	0.78
Total cost of briquettes containing 1 ton metallic iron,	$7.41

We have now to consider the repairs to the furnace and number of charges made per 24 hours. The average total duration of a test-heat, as previously stated, was 8 hours, 50 minutes, while the corresponding time in the ordinary pig-and-scrap process is about 8 hours; this would make a difference of about one hour in the time required.

Now in the ordinary pig-and-scrap process, the amount of gas-coal consumed amounts to about 30 per cent. of the weight of steel obtained. Consequently, if the charge in the Imperatori process were to last about one hour longer, it will require about $\frac{1}{8}$ of coal more, or say in all about 34 per cent. of the weight of steel.

As to repairs, it has been already stated, and with the ordinary so-called ore-process it has been demonstrated at many works in Europe, that a neutral lining will require very little repairs and will last for several hundred heats; so that the expense of re-lining can be considered to be the same in both cases.

The same might be said of the number of heats in 24 hours. This number, as is well known, depends largely on the repairs required by the furnace. As these repairs will not be heavier in one case than in the other, we may say that practically the same number of heats can be made in both cases in 24 hours.

If we admit, furthermore, that the loss of iron in the pig-and-scrap process is about 10 per cent. and that this loss in the Imperatori process will not be higher, as has actually been shown for this latter process, by several heats during the tests (and it is even lower than 10 per cent. according to the tests made in Europe), we can easily establish, as to the cost of steel in the two processes, a comparison which may be confined to those items which constitute the difference between them.

Ordinary Pig-and-Scrap Process.

One ton of iron charged { Pig-iron, 0.30 @ $19, . . .	$ 5.70
{ Scrap, 0.70 @ 21, . . .	14.70
Loss 10 per cent.; hence, cost of 0.9 ton product, . . .	$20.40
" 1 " " . . .	22.67

Imperatori Process.

0.40 ton pig-iron, at $19, ...	$7.60
0.20 scrap, at $21, ...	4.20
0.26 iron in 40 per cent. ore, at $7.41, ...	2.96
0 86 ton iron charged, ...	$14.76
Hence, 1 ton of iron charged (or, after allowing for 10 per cent. loss, 0.9 ton of product), would cost, . . .	$17.16
Or, cost of 1 ton of product, ...	19.06
To which must be added 4 per cent. excess of gas coal over the 30 per cent. of the weight of the steel which is required in the ordinary process, or 0.04 ton, at $3, .	0.12
Making a total, for material and fuel, of	$19.18

The difference in favor of the Imperatori process is $3.49 per net ton of steel obtained. This economy is important, but it does not constitute the only advantage of the process. In an average charge in the Imperatori process, composed, as has been said, of 40 pig, 20 scrap and 40 ore, the chemical composition of each constituent can be accurately ascertained. The pig-iron will generally be low in silicon, sulphur and phosphorus; such iron can easily be obtained at reasonable price. The scrap used can be wholly supplied by the

crop-ends, rail-ends, etc., from the works themselves, no outside scrap being needed; consequently its chemical composition will be the same as that of the finished product, and can always be relied upon.

As to the ores used in the briquettes, there will be no difficulty in securing, at low rates, concentrated ores (not much used at present) almost free from phosphorus and sulphur, and consequently constituting an excellent material for the production of the best quality of steel.

The carbonaceous material for the briquettes may be either coal or coke or charcoal or anthracite, as long as it does not contain too high a percentage of sulphur. Charcoal is very well adapted to this process, as it is very generally free from sulphur and phosphorus. Thus the well-known chemical composition of the raw materials used will allow of obtaining, at a low price, a very good quality of steel, which is not always obtainable in the ordinary pig-and-scrap process.

Another very advantageous feature of the Imperatori process is the facility with which many ores, hitherto almost useless, could be treated, thus enlarging their market, now very limited. Thus, many lean ores largely found in the States of New York and New Jersey, and, in fact, in almost every one of the United States, would constitute, when once ground and concentrated, an excellent material for this process.

In Morris county, New Jersey, ores are mined containing only from 30 to 37 per cent. iron, with 0.085 sulphur, and 0.06 phosphorus, which, concentrated, furnish powdered ore containing 64 to 65 iron, 0.02 sulphur, and 0.02 to 0.03, or even less phosphorus. These are well adapted to be used in the Imperatori process.

Besides these concentrates, most of the magnetic sands found in large deposits in Canada and in the United States, and containing a high percentage of iron (as high as 66 per cent. in certain cases), could be made available by this process. Many of these sands, if not rich, are physically associated with earthy substances which a simple washing could remove, furnishing a practically almost pure oxide of iron in powder, at a low price and particularly well adapted to be made up into briquettes.

Another advantage of this process is its easy adaptation to any existing open-hearth plant, as it can be carried on in any existing furnace without the slightest alteration or addition.

THE MOUNT MORGAN MINE, QUEENSLAND.

BY T. A. RICKARD, MELBOURNE, AUSTRALIA.

(Cleveland Meeting, June, 1891.)

AMONG the gold-deposits discovered in recent years none is more extraordinary in richness or interesting in structure than that of the famous mine at Mount Morgan. At a time when but few Australian mines were known to the world outside the colonies, Mount Morgan was quoted as an occurrence, unusual not only in its origin (for it was said to be due to geyser-action) but also in the purity of its gold.

The mine is situated just within the tropics, twenty-six miles southwest of Rockhampton, in central Queensland. Queensland attained in 1889 the first place among the gold-producing colonies of Australasia, a position previously always held by Victoria. In that year Queensland produced 737,822 ounces, while Victoria came second with 614,838 ounces. Of the four chief mining districts Rockhampton stood first, as the following figures show:

	Quartz crushed, Tons.	Yield, Ounces.	Average per ton, Oz.	Dwt.
Rockhampton, ...	81,138	340,669	4	4
Charters Towers, ...	109,328	165,551	1	10
Gympie, ...	106,625	112,847	1	1
Croydon, ...	29,423	51,156	1	15

It should be added that Queensland has but little alluvial mining, the total output from this source in 1889 being only 5044 ounces, none of which came from the four principal gold-fields. Leaving out Rockhampton, the output of which is practically that of the Mount Morgan mine, it will be noted that the more northerly gold-fields show the highest average (Gympie being furthest from and Croydon nearest to the equator). This is largely explained, however, by the increased cost of milling, due chiefly to the want of water for batteries and the consequent economical necessity of selecting high-grade material only for crushing.

During the year ending November 30, 1889, the Mount Morgan

mine produced 75,415 tons, yielding 323,542 ounces, 13 dwts., 13 grs., worth £1,331,484 18*s.* 5*d.* (about $6,657,424), while the expenditure was £227,769 19*s.* 8*d.* (say $1,138,849) permitting the payment of £1,100,000 ($5,500,000) in dividends. The yield per ton was 4 oz., 6 dwts., 4 grs., while the working-cost was, as is seen, only 17 per cent. of the value of the product. These figures speak for themselves.

So rich a mine would be expected to have some romance woven about the story of its discovery. Numerous and various tales are told of the first recognition of its value, but the best authenticated facts are as follows: The property consists of the original selection (No. 247) of 640 acres taken up for grazing purposes, in 1873, by Donald Gordon. Becoming acquainted with the brothers Morgan, who also held land in the district, he showed them one day a piece of gold-bearing quartz which he had picked up in Mundic creek. For a consideration, stated to have been £20 and as much whiskey as he could drink, Gordon agreed to indicate to them the locality of the find. On the hill overlooking the creek he showed them the siliceous ironstone, some of which can still be seen cropping out on the north-eastern slope. The stone carried visible gold; they found by sending samples to Sydney that it was even richer than they had imagined; so they purchased Gordon's holding at £1 per acre.

The three Morgans subsequently sold, first a part, and eventually the whole, of their interest in the mine. In 1886 a company was formed with a capital of one million shares of £1 each. These shares rose toward the end of 1888 to £17 5*s.* (about $86.25) giving the mine a market value of seventeen and a quarter millions sterling or over eighty-six million dollars. The shares are now quoted at £7; for Mount Morgan, to quote the language used by the managing director at an annual meeting, "is after all only a gold-mine and is consequently subject to the vicissitudes of all mineral formations," which is a truism too often forgotten by those who conduct mining operations.*

Description of the Mine.

The mine does not, as its name would imply, crown the summit of a mountain, properly so called, but forms a quarry at the top of a

* Of the neighboring companies (and as might be expected they are numerous, with their suggestions of an extension of the ore-deposit in such names as Mount Morgan West, Mount Morgan Extended, Mount Morgan North Consols and so on *ad nauseam*) none have proved profitable, notwithstanding the expenditure of much money in numerous and scattered trial-shafts, tunnels, etc.

hill, only 500 feet above the village at its base and 1225 feet above sea-level, surrounded by very broken hilly country and almost encircled by a small stream (the Mundic creek already mentioned), and in many respects distinct in position and geological structure from the hillocks about it. From the summit can be seen the level line of the "desert sandstone," crowning the highest ridges of the neighboring hills—spurs from the main range which under different names (Blue Mountains in New South Wales and Australian Alps in Victoria) traverses the three colonies near the east and south-eastern coast of the Australian continent. The base of the sandstone is slightly lower than the summit of Mount Morgan and overlies greywacke and quartzite. Dykes, of at least two periods, form an important feature of the structural geology.

The crest of the mount is being rapidly broken away in the quarrying operations which have supplied the great output of the mine. The removal of from 1200 to 1700 tons a week makes a big hole in a year. While the greater part of the summit has been thus removed, the northeast slopes still show the croppings which first excited Gordon's attention.

The ground is worked in terraces or benches, 30 feet high and 200 feet long. At the date of my visit five of these, started at various times, could be seen, though but little remained of the floor, 25 feet from the top, which formed the open cut first put into the ore. A central shaft passes from the floor of the second terrace and connects with the deeper tunnels, a series of which have intersected the deposit in various directions. This shaft, at 206 feet, connects with the Freehold tunnel, the main ore-way of the mine, which is 789 feet long. The next deeper tunnel, called No. 1, starting at right angles to the Freehold, from the southern face of the hill and penetrating it for 1070 feet, is 155 feet below the floor of the lowest surface-working and 320 feet below the original summit. This No. 1 tunnel, though only 33 feet deeper than the Freehold, affords a very interesting section of the mountain and indicates a great change in the form of the ore-deposit. It is the deepest of the adits (for adits and not strictly tunnels most of them are), except the "Sunbeam," which had been started at the time of my visit; but it had not been advanced sufficiently to throw any additional light upon the structure of the mountain.

The Mount Morgan ore is remarkable for its extremely heterogeneous character. The frequent alleged discoveries of "a second Mount Morgan," supported by similarity of the specimens exhibited

to specimens of the Mount Morgan ore, are not to be wondered at, in view of the great difference in appearance of fragments broken in different parts of the same heading of this mine. The material quarried in the upper workings, samples of which are now to be found in collections and museums all over the globe, is generally so friable and shattered, as to render its removal with the aid of black powder only, very easy. Standing in one of the open cuts, one can see faces of bluish-gray crushed quartz very similar to Comstock ore; masses of siliceous hematite (usually considered the most typical Mount Morgan stone) resembling the croppings of ordinary gold-veins, whether in Victoria or California; heavy black iridescent iron-stone which might be the cap of a silver-lode; and light-colored reddish ore which might come from the gossan of a copper-mine. Some of the material is crushed to the consistency of sugar, while in other portions of the mine men are seen employed in breaking boulder-like masses of ore. Stalactitic forms occur in cavities, whilst a vermicular and reniform appearance is also not uncommon. Very rich returns were obtained from a body of bluish-black, beautifully iridescent ore. The Freehold tunnel-workings encountered a patch of very pure white, porous, friable "sinter," so light, owing to the imprisoned air, that it would float on water.

Mount Morgan is not a "specimen-mine," though in its early history it furnished very lovely pieces of gold-quartz, some of which I saw in Sydney. They consisted of an iron gossan, containing big "splashes" of gold, of the size of a thumb-nail. As a rule, however, the gold, even when visible, is very fine and scattered thickly through the stone; but owing to its peculiar character (the coating of oxide of iron to be referred to later on) it is not readily detected by the eye.

Examination of the many varieties of ore shows that while there may be a great difference in outward appearance, due to coatings of many-colored oxides, the ore is always substantially quartzose. As seen in the surface-workings, the deposit may be considered a mass of quartzose material of varying color and specific gravity, traversed by a series of dykes having a general N. W. and S. E. direction. Its extent is approximately indicated by the various workings which intersect the hill. In the upper part of the mine, while the north and northeastern slopes prove it to be continuous in these directions, its eastern limit has been reached in the lower floors or benches; and to the southwest it is bounded by a large felstone dyke, which forms the most marked feature of the surface-excavations. The first

tunnel to pierce the hill is No. 2, which starts from the fifth or lowest floor, running N. 10° E. and penetrating 120 feet of ore before it reaches a dyke. This is probably the dyke above mentioned as the boundary of the deposit in the upper floors; but here it has evidently cut into the deposit. Beyond this dyke the tunnel is in ore for 200 feet further, before meeting another intrusion of felsitic matter, which extends for the remaining 130 feet to daylight. A branch-tunnel proves the continuity of the ore-body eastward. At this level the stone evidently holds out most satisfactorily, and is, I believe, for the most part auriferous. It is a porous siliceous material, varying between a light "sinter" (sometimes white, sometimes iron-stained) and a heavy iron-stone, often manganic, the latter in boulder-like masses.

The next tunnel is the Freehold, forming, as has been said, the main artery of the mine. It connects with the shaft (sunk from the No. 2 floor), and from the mouth of it a tramway carries the ore to the chlorination-works. The tunnel starts from the southeast face of the hill, traversing a decomposed rhyolite for the first 180 feet, and then cuts through 40 feet of pyritiferous quartzite, a rock which here first makes its appearance and which will be seen to play an important part in the geology of the locality. Leaving the quartzite, the tunnel cuts through 180 feet of a rock which I recognize* as a normal dolerite. The innermost portion is decomposed, and abuts against a much altered feldspathic rock. This brings us to 390 feet from daylight; the remainder of the tunnel (397 feet) traverses the ore-deposit. The ore is a light sinter-like quartz, often iron-stained; but while it contains short rich patches the general tenor is low. Near the junction with the main shaft, almost at the end of the tunnel, a branch cross-cut runs southward for 237 feet. The first 186 feet are in the deposit, which here also is poor, though in appearance it does not differ from the richer portions.† The next 20 feet traverse the dyke which at this point, as in the upper floors, forms the limit of the ore-body. Beyond this there is only to be seen the pyritiferous quartzite, which also appears in a short cross-cut to the east.

The next deeper tunnel is No. 1, only 33 feet below the Freehold. I was denied access to this, and am, therefore, indebted to a govern-

* Thanks to microscopic sections shown to me at Charters Towers by Mr. Clarke.

† It is here that one sees the light, white, pumice-like material which, owing to the air in its cavities, will float on water.

ment report* for the following particulars: This adit, 1070 feet long, starts in, and traverses, the quartzite of the country for 132 feet before passing through a narrow dyke; it then passes through 67 feet more of highly pyritiferous quartzite, which continues to within 12 feet of a shaft coming down from floor No. 5. There next succeeds a large felstone dyke (the one noted as cut in the tunnel above), and the remainder of the adit continues in pyritiferous quartzite, alternating with numerous dykes, until the last 200 feet are reached, which are occupied by an altered dolerite. The auriferous material of the upper workings was represented in this adit by 25 feet only of siliceous iron-stone. Two branch cross-cuts similarly prove the absence of the deposit in this part of the mine at this depth, as they also intercept the quartzite of the country.

A still lower tunnel, the Sunbeam, starts from the west face of the mountain, passing through pyritiferous quartzite; but it was not advanced sufficiently at the time of my visit to afford any evidence as to the extent of the ore-deposit.

It will be noted that the largest section as yet obtained of the auriferous portion of the deposit is in tunnel No. 2, where its dimensions are 356 feet (26 feet of which is occupied by the big dyke) in a north direction, by 310 feet east, as proved by a branch cross-cut. The deposit has been proved to extend further eastward in the deeper levels (Freehold tunnel) than it did in the surface-excavations, while on the other hand its western limits are more restricted.

Theories of the Formation of the Ore-Deposit.

The origin of the ore-deposit has been, as might be expected, the theme of much controversy. The earliest description came from Mr. R. L. Jack, the head of the Queensland geological survey, well known as a careful observer. His report was made officially for his government and appeared in 1884. Others have contributed their opinions since; and meanwhile the rapid development of the mine, more particularly by the deeper adits, has furnished additional data for a problem, of which the following solutions have been offered:

1. That the deposit is that of a geyser (R. L. Jack).
2. That it is an auriferous zone traversed by a series of quartz veins of auriferous mundic (J. Macdonald Cameron).
3. That it is the decomposed cap of a large pyrite-lode (the view held by several local and other mining engineers).

* R. L. Jack's second report, which will be referred to again.

The Geyser-Theory.—The first of these explanations, known as the "geyser-theory," was promulgated by Mr. Jack in his first report, dated November 8, 1884. It has found its way into many scientific publications, and has been until recently the most widely accepted. Citing as a similar occurrence the hot springs and geysers of the Yellowstone Park, described by Dr. A. C. Peale in the United States Geological Survey reports, Mr. Jack observes in his first report: "Nothing but a thermal spring in the open air could have deposited the material under consideration." The two sections shown in Figs. 1 and 2 are taken from that report. Of Fig. 1, he says: "The above diagram represents my idea of what would take place in the case of a geyser remaining in activity for a (geologically speaking) lengthened period." The original form of the ground is shown by *aa*, the deposit of precipitated material by *bb*, the layer of solid material at *cc*, while the surface-contour is indicated by *dd*. In describing this section he concludes, "Such, I believe, is the history of Mount Morgan as we now see it."

The structure of the mountain is supposed to be shown by Fig. 2, in which Mr. Jack indicates rhyolite dykes at *d* and metamorphic rocks at *s*. He says: "*a* is the pipe of the geyser (theoretical), *b* the cup-deposit and *c* the overflow of the geyser."

These two sections have found their way into scientific publications all over the world, and have been for a long time accepted as most interesting explanations of one of the richest of modern gold-mines. But they evidently are altogether imaginary and theoretical.

The same geologist says, in his description: "After the cessation of thermal activity the powers of sub-aërial denudation would come into play. Denudation would obliterate the lateral terraces which were probably not absent from the slopes of Mount Morgan." The information obtainable in 1884 as to the nature of the deposit was totally insufficient to enable an observer, however careful and experienced, to build a theory upon. The amount of work done at that time was mostly limited to one open cut, 10 by 15 feet in size, which showed a fan-like arrangement of the ore, suggesting to Mr. Jack the structure of a geyser. It was an altogether local appearance, as was proved by the work of a few days later.

In a second report, dated December 12, 1888, Mr. Jack adds: "The evidence now to hand, in my opinion, goes far to confirm my original view that the auriferous material was deposited by a thermal spring." The deposit had been asserted in the first report to have been formed "in the open air," a condition constituting the whole

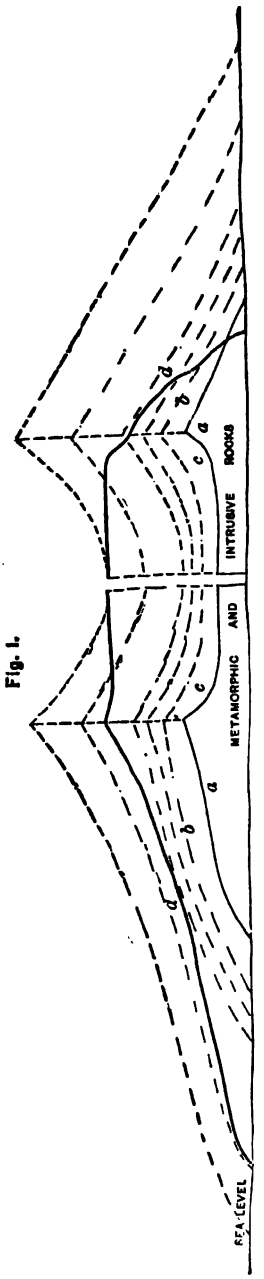


Fig. 1.

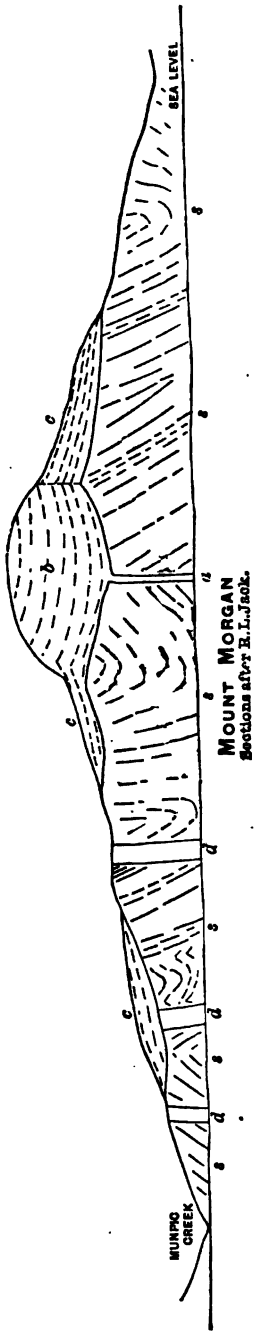


Fig. 2.

MOUNT MORGAN
Sections after R. L. Jack.

distinctive value of the theory, which otherwise might not differ materially from the views accepted as to the aqueous origin of ore-deposits in general. In the second report the author declares his theory to have been confirmed, but produces evidence which really contradicts its essential feature.

Taking the two reports together, we find that, in support of his explanation, Mr. Jack adduces: (*a*) the "fan-like arrangement" of the material; (*b*) "the frothy and cavernous condition" of the siliceous sinter; (*c*) the hydrated condition of the silica and (*d*) the fact that one of the dykes intersecting the country does not also intersect the deposit.

(*a*) The "magazine" face, at the time of Mr. Jack's first visit, appeared to him to have a distinctly fan-like structure. But little work having been done since that time at this particular point, it should be possible to verify this statement; but the most careful observation does not confirm it. The cutting shows an arrangement of loose material such as can frequently be seen in surface-excavations, probably less marked now than when first seen by Mr. Jack; but the entire absence of any repetition of such an arrangement of material in the other workings must destroy its value as evidence, if it ever had any.

(*b*) The "frothy and cavernous" condition of the quartz is not peculiar to Mount Morgan. Those who are familiar with the pocket-mines of California will remember the very light, pumice-like quartz which often accompanies the gold in those mines. It is the light honeycombed quartz found in the gossan of gold-veins in many places. Generally speaking, the Mount Morgan gangue can be duplicated elsewhere; and the particularly light "frothy sinter," noticeable as a patch in the deeper workings, differs only from the bulk of the quartzose ore in that it is remarkably free from accessory minerals. It is the siliceous skeleton produced by surface-decomposition of more than ordinary intensity. It is possible to produce artificially in a few minutes, by the application of acid to very dissimilar iron-stone ore, a material similar to that noted, which has undergone the chemical action of underground waters during long periods.

(*c*) The hydrated condition of the silica is not unusual in quartz-lodes, and while it is valuable as evidence to disprove the suggestion that the ore is the replacement of anhydrous quartzite, it does not necessarily point to the agency of a geyser.

(*d*) The intersection of the country by a dyke which does *not* pene-

trate the deposit is an important fact. The dyke referred to is that cut by tunnel No. 1 at 120 feet from the shaft coming down from floor No. 5. Mr. Jack says, at the end of the second report: "This shows that the sinter and iron-stone are deposited on, and were not altered portions of, the pyritous quartzite country-rock." Unfortunately the evidence is far from conclusive. In describing No. 1 tunnel the dyke is referred to in the following sentence: "A quartzose rock full of fine pyrites is traversed for the next 37 feet, when a dolerite or rhyolite dyke is cut. The direction of this dyke is uncertain, as the tunnel is here timbered up." In his descriptions Mr. Jack always endeavors to draw a marked distinction between the "felstone" dykes which penetrate the ore and the "dolerite" and "rhyolite" which intersect the country-rock. Now, this distinction is arbitrary and misleading. I examined the rocks both *in situ* and, a few days afterwards, under the microscope. The dykes penetrating the deposit are so decomposed that it is impossible now to determine which is felstone and which rhyolite. The dykes cutting through the country are similarly, but not to so great a degree, decomposed, especially in the vicinity of the deposit. The statement that a "dolerite *or* rhyolite" dyke does not penetrate a deposit which is freely intersected by decomposed feldspathic eruptives, should be supported by proof. It is not so here; for the dyke cut in No. 1 tunnel has not been followed upward, and in the south branch of the Freehold tunnel, just overhead, there is a dyke, which is possibly the one in question.

My sections, shown in Figs. 3 and 4, will illustrate this. The left half of Fig. 4 resembles one of Mr. Jack's sections, being taken along a nearly identical plane (mine is taken due north and south). His section, however, gives a curved contour to the limit of the deposit, very nicely, but unwarrantably, suggesting the shape of a geyser-basin. The right half of the section is not filled in, since this part of the mount has not been thoroughly developed. Fig. 3 is taken along a line E. 17° S., and is obtained by projecting the cross-cuts from the No. 1 and Freehold tunnels upon the plane of the No. 2 tunnel.

In further considering the conditions under which thermal springs and geysers* exist, and the similarity of such conditions to those obtaining at Mount Morgan, it may be noted:

1. Isolated geysers are unusual, nothing being more remarkable

* A geyser may be defined as a thermal spring ejecting material in the open air.

than the extended area of such phenomena. But vigorous and extensive prospecting, such prospecting as always follows great mineral discoveries, has not led in this region to the finding of anything

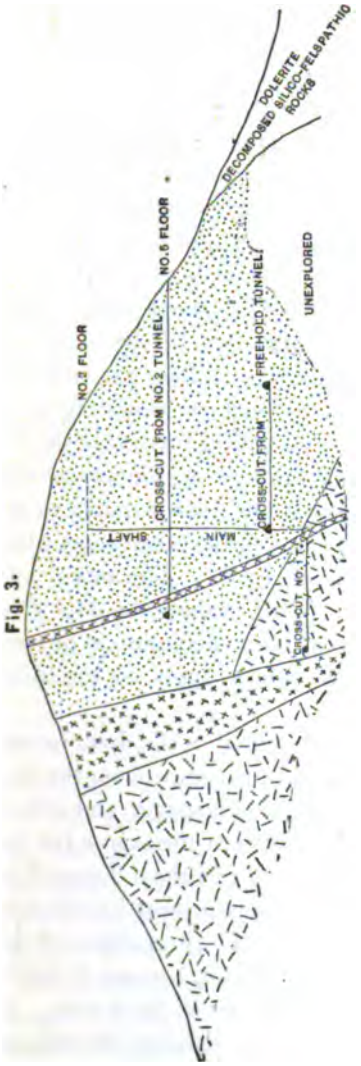


Fig. 3.

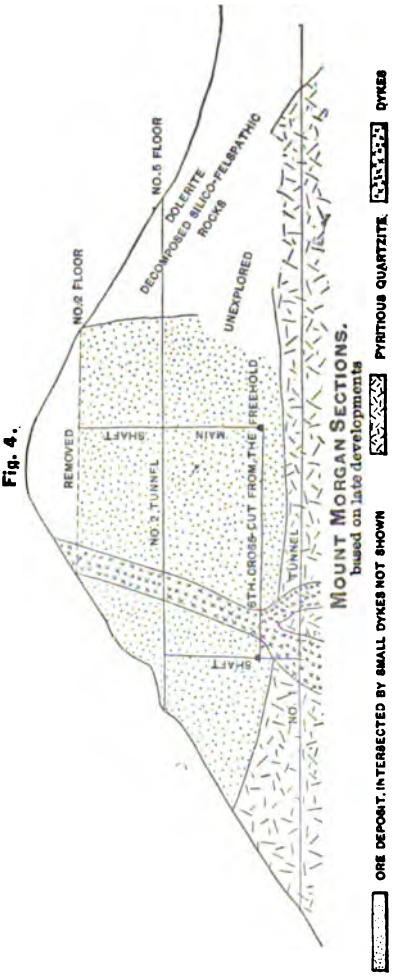


Fig. 4.

similar to Mount Morgan or to a geyser. It is true, there is a small hillock adjoining the mount to the northwest (called Callan's Knob) where somewhat similar material has been found; but it may be considered as an offshoot or spur from the main deposit.

2. Geysers have terrace-formations due to the overflow of the siliceous or calcareous waters. The map accompanying Mr. Jack's first report indicated "the overflow from the hot spring" as covering an area of 2000 by 3000 feet. But this was purely a guess, for such overflow-material is not to be seen; had the terraces existed, and had they been removed by denudation, would not the adjoining river beds show some accumulations of their *detritus?*

3. Geysers have a central vent, which, when the geyser becomes extinct, is found to be choked up with sinter. Though Mount Morgan has been traversed by a number of adits and cross-cuts, not to mention extensive surface-excavations, no such vent and no such central column, or anything suggestive of either, has been seen.

The Network-Theory.—The second explanation is that of J. Macdonald Cameron, namely, that it is a network of quartz veins. This opinion was expressed in a report to the directors of the company, dated March 26, 1887. The following passage outlines the author's ideas: "It (Mount Morgan) may be considered as consisting of a net-work of quartz veins, about 200 feet in width, traversing on the one hand a metamorphic matrix of a somewhat argillo-arenaceous composition, largely impregnated where it has been exposed to atmospheric influence with oxides of iron, and on the other hand what appears to be a felspathic tufaceous igneous rock." The author advances no proof that such is the case. What evidence he does give is couched in the vaguest phraseology. This explanation is proved untenable by the same reasons as those which apply to the third theory now to be considered.

The Lode-Theory.—The view that this deposit is the decomposed cap of a pyrites lode, has derived fictitious importance from the fact that it is alleged to be the opinion of the mine-manager and others interested in the property. Such a view has been the excuse for the large number of "extended" companies (they call them "pups" in Victoria), companies which, in the light of our present knowledge, were almost certainly foredoomed to failure. The prevalence of the lode-theory in the early days of mining at Mount Morgan is not to be wondered at, however little warrant there may be for it now. It was due to the existence of mundic veins in the vicinity, the casts of pyrites crystals found in the ore, and the neighborhood of the pyritiferous quartzite, the true relation of which was not known, or if known, was kept very quiet. The well-marked limits of the great deposit, as outlined by the deeper developments of the last two years, completely disproves this theory and that of Cameron. No doubt,

the assertion of the continuity of the gold-bearing rock, even though it should become pyritiferous, has fascinations for a shareholder which even a geologist can understand. But the deeper tunnels have proved that lode-structure is entirely absent, and that the deposit lies upon, but is not an altered portion of, the pyritiferous quartzite of the country.

The Theory of Metamorphosis and Replacement.—The more than ordinary size and value of the Mount Morgan ore-deposit, the romantic history of its discovery and its subsequent marvellous production, have all helped to place it outside the ordinary type of mineral occurrence. This is often the case with new discoveries, and it is not the first time that the geyser has been a *deus ex machina* to a perplexed geologist. The splendid work done in recent years by Sandberger, Daubreé and others in the study of the chemical change, and the distribution by underground waters, of the minerals of the older rocks has led to a gradual substitution, for theories of eruption, or direct igneous agencies in general, of the belief in the capability of aqueous agencies to bring about most of the phenomena of ore-deposition. Moreover, the action of percolating solutions on the minerals of the crystalline rocks, and subsequent interchange with the more soluble portions of the neighboring rock-masses, have, to a great extent, replaced the assumptions of the old ascension-theories. The view that the more ore-deposits are studied the more frequently they will be found to be in great part the product of the replacement of the country-rock is gaining ground, and has been much strengthened by the researches of Becker at Virginia City, Curtis at Eureka, and Emmons at Leadville, as contained in the monographs published by the U. S. Geological Survey, monographs (the last-mentioned, particularly), which are the classics of modern economic geology. This view, which underground experience is continually confirming, has explained many of the more unusual types of ore-deposition in Europe and America, and I believe it will apply to Mount Morgan.

The explanation suggested is this: that the ore-deposit of Mount Morgan represents an altered portion of shattered country-rock, which, by reason of its crushed condition, was readily acted upon by mineral solutions, and that these solutions replaced the basic and feldspathic with acidic and quartzose material, which was also gold-bearing. It is its quartzose and permeable character which has saved from disintegration the mass thus affected, and has preserved it as an ore-body on the summit of the hill.

A case, in many respects very similar, occurs in the Red Mountain basin, Colorado, and the following description given by Mr. Emmons* is clearly in point:

"Instead, however, of being surface-deposits, they [the Red Mountain deposits] are simply portions of the andesitic country-rock from which acid waters have removed the basic constituents, perhaps depositing a certain amount of silica in their place; the resulting quartzose masses, offering greater resistance to the disintegrating effect of atmospheric agents and to erosion than the surrounding rocks, have been left as mound like ridges protruding above the general surface, more or less independent of the natural drainage-channels."

This explanation, curiously enough, was also given to disprove a geyser-theory, but the deposits in question seem to possess scientific interest only.

Deposits of somewhat kindred origin are the Yankee Girl and Bassick mines in Colorado. The former, a well-known mine in the San Juan district, is perhaps the more suggestive, since in that case several fracture-planes meet to form a wedge-shaped mass of rock, which has been so acted upon by percolating waters as to become a "chimney" of very valuable silver-ore.

To return to the particular conditions which obtained in the district under discussion: The country around Mount Morgan consists, in descending series, of sandstone, greywacke, quartzite, shales and occasional bands of serpentine. Intrusive masses of dolerite interpenetrate the country, and often occupy a position between the originally stratified rocks. The sandstone is known in the geology of Queensland as "Daintree's desert sandstone," and, by virtue of a recent discovery of fossil remains, may be labelled as Cretaceous. The greywacke has not been recognized as such on the Mount itself, but it underlies the sandstone in other parts of the district. It is a mixture of silico-feldspathic material, which shades off into a greywacke-slate, and is usually so much altered as to be scarcely distinguishable from a true eruptive. The quartzite has a bluish-gray color, and is highly pyritiferous, the pyrites carrying traces of gold.

The shales are usually so indurated and altered as to resemble closely the crystalline rocks. The dolerite, when fresh, is a typical rock of its class, and in various intrusive forms is cut through by most of the prospecting-shafts at the base and around the mountain. The serpentine is of very limited occurrence. Some uncertain, undetermined decomposed rocks, which have also been called dolerite,

* "Structural Relations of Ore-Deposits."—*Transactions*, xvi., 809.

are probably metamorphosed greywacke. The country is freely intersected by dykes of varying composition, especially near the ore-deposit.

The history of these rocks I read somewhat as follows: A period of dynamic disturbance is indicated by the intrusions of dolerite, which, by extreme metamorphism, might have changed a dolomite into the serpentine we now see; would have indurated the shales so that they are scarcely to be distinguished from the crystalline rocks; and would also, accompanied by chemical alteration, change a ferruginous red sandstone into a bluish-gray, highly pyritiferous quartzite. Approaching the surface, the same energy would be expended in the fracturing of the quartzite and greywacke; the intrusive dolerite would rise through the fissures in the shattered rocks, forming dykes, which, meeting a silico-feldspathic granular rock (the greywacke), would give it a semi-crystalline character. The sandstone would similarly be vitrified. Later movements would result in the further intersection of this part of the district by the numerous dykes, the decomposed remains of which are now to be seen ramifying through the deposit. Those gradual chemical interchanges would take place which resulted in the alteration of the shattered country-rock, and its becoming a portion of the gangue inclosing the auriferous material, which was then, or at a later time, deposited. In process of time, subaërial denudation removed the sandstone, which now is only to be seen on the further summits of the neighboring hills. Atmospheric agency continued to carve away the less siliceous and less porous portions of the country surrounding the deposit, until Mount Morgan, owing to the pervious, quartzose nature of its crest, remained as a low hill in an undulating country.

The district of Mount Morgan has undergone extreme metamorphism, accompanied, as such metamorphism frequently is, by those chemical reactions which more or less completely alter the rocks acted upon.

It is probable that there was more than one period of such activity, alternating with intervals of rest, during which the pressure of overlying rock-masses and the cementing action of the underground waters would slowly re-solidify the more or less shattered rocks.

The deposit occupies a shattered portion of country-rock, freely intersected by dykes. The largest dyke in the upper workings cuts through the ore at one level, while it bounds it at another. It seems in keeping with experience elsewhere, to suppose that the dykes

penetrating the country formed passages along the line of contact, through which and around which, by the agency of mineral solutions, the deposition took place, sometimes on one side only, sometimes on both, the physical condition of the different portions determining, in a large measure, the extent of such deposition and replacement. It is also probable that, after the formation of the deposit, later movements caused the circulation of fresh mineral-bearing solutions, which would lead to the enrichment of the deposit at certain points, and so explain the irregular tenor of the ore.

The evidence at hand is not sufficient to enable us to determine the particular rock in which the deposit was formed, and which it almost entirely replaced; but it could not have been the underlying quartzite, the line of contact with which shows no gradation into the ore; and of the other two possible rocks, the greywacke and the sandstone, the indications point to the former.* In the neighboring prospecting-shafts, auriferous quartzose material resembling that of Mount Morgan has been found to pass into a greywacke or greywacke-slate, which, in turn, overlies the pyritiferous quartzite. Silicate of alumina is generally present throughout the deposit. The character of the ore-material is such as would result from the alteration of a silico-feldspathic rock by the action of mineral-bearing solutions.

The numerous dykes which penetrate the deposit are the evidence and the result of the dynamic agency which fissured the country-rock, and in so doing prepared a ready passage for the underground waters. These waters, while they may have been "thermal" (*i. e.*, of a temperature above the mean annual temperature of the district, as are most underground waters), yet did not undergo precipitation "in the open air." The brecciated forms which the ore sometimes assumes scarcely require the explanation offered by Mr. Jack, who states that "the occasionally angular condition of the sinter and tumbled condition of the ironstone masses, would appear to indicate that explosive discharges of gases or steam occurred at intervals with sufficient violence to disturb the deposits accumulated by the thermal spring." Would it not be simpler to suppose the fracturing of the rock, of which there is abundant evidence, to have produced irregular pieces, which would subsequently become partially rounded

* The position of the greywacke has been shown to be between the quartzite and the sandstone. The original summit of the Mount is but little below the level of the base of the sandstone, while the deposit, as we have seen, overlies the quartzite.

by the solvent action of the waters percolating through their interstices?*

At Mount Morgan all the conditions generally considered favorable to ore-deposition were present in a marked degree. A mass of country-rock, whether greywacke or sandstone, consisting of granules of quartz, held together by a feldspathic cement,† becomes fractured by the intrusion of a series of dykes, which form a ready passage for the flow of mineral solutions. Such a rock would readily lend itself to alteration, and would be well fitted to receive a mineral deposit. Later dynamic action produced a further metamorphism of the surrounding rocks, followed or accompanied by the intersection of the already shattered rocks by another series of dykes, which reopened a passage for the underground waters. In this case it is not necessary to go far for the source of the gold. The large mass of decomposed pyritic quartzite, though the pyrites contains but a trace of the precious metal, is more than sufficient to account for the wealth that has been uncovered.†

This theory explains several points which have been the source of perplexity, more particularly the want of regularity in structure, which was as much opposed to the geyser- as to the lode-theory. The great unevenness of the gold-contents, the frothy state of the quartz, so often noted, and also its hydrous condition, may be due to

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* Since writing the above, I have visited and examined the equally famous Broken Hill silver-deposit. When I declared that the material of the Mount Morgan ore was not unique, that it could be duplicated elsewhere, and that it did not, *per se*, require any extraordinary explanation, I did not expect to see so soon an instance confirming my statement. The huge outcrop of the Broken Hill silver-lead lode has much to remind one of the big gossan of the Queensland gold-mine. Indeed, if one stands in the quarry between Brodribb's and Patterson's shafts one might easily imagine oneself on floor No. 3 of Mount Morgan. The face has the same singular, broken appearance, consisting of masses of ochreous earth, bands of hard, siliceous ironstone, bodies of black manganic ore, together with portions of white kaolinized material (here undoubtedly altered inclusions of country-rock, and not, as at Mount Morgan, decomposed felstone dykes). Both at the surface and underground, the sintery stalactitic structure which is found at Mount Morgan also occurs. The hydrated iron-ores produce the same "frothy," "cavernous" material, supposed to be so suggestive of a geyser—and this, too, not in isolated patches only, but widespread. I may add, that there is no doubt whatever that the Broken Hill is a "true fissure"-lode, traversing the metamorphic rocks of the district (schists, garnetiferous sandstone, and quartzite, principally the first), independent of their stratification.

† The underlying quartzite may not be the only source of the gold, for it is noteworthy that the "desert sandstone" has been known to be auriferous, such gold being waterworn and sedimentary.

later precipitation in cavities already formed by the solvent action of the waters on the shattered portions of rock. Finally, there is no need to offer any particular reason for the unusual richness of the deposit, beyond the eminently favorable physical condition of the original rock, the numerous channels offered to percolating waters and the close proximity both of the source of the gold and of the usual precipitants.

Whether this be a correct explanation of the facts, time and the further development of the mine will probably decide. It may certainly be said, without fear of contradiction, that few mines offer such an interesting field for geological speculation.*

Treatment of the Ore.

The Mount Morgan chlorination-works are among the most successful and extensive now in operation. The complete success of the treatment is largely due to the extremely friable character of the ore, which renders its pulverization easy, while its porosity assists materially in the thorough chlorination of the gold. An enthusiastic writer has spoken of the ore as "a sort of snow-drift, which melts in the chlorination-vats of the company into a golden sand, such as might be supposed to have been brought from the bed of the river Pactolus, instead of from the top of an Australian mountain."

The capacity of the present works is 1800 tons per week. The gradual alteration and enlargement of the method of treatment is shown by the stamp-mill and the two chlorination-plants. The mill (25 stamps) by the side of Dee creek, is a reminder of early attempts at the extraction of the gold by ordinary amalgamation. The rock which the battery was called upon to crush averaged 10 ounces

* Since I wrote the above there has come into my hands the report of the Mount Morgan Gold Mining Company, Limited, presented at the last annual meeting, and giving information up to July, 1890. It is accompanied with maps and sections, notably calculated to yield a minimum of information. It appears from the report that later developments have added but little to the known extent of the ore-body, the main feature of the later exploration being the driving of a cross-cut south from the Freehold, at 800 feet from its mouth, proving the extension of the deposit at this line and in this part of the mine—as was to be expected, seeing that its dip is toward the mouth of the Freehold.

Up to November 30, 1890, the mine had paid the present company £2,358,333 in dividends, the total gold obtained being 756,042 ounces, worth £3,121,741. During the last six months (ending November 30, 1890) 37,867 tons of stone were treated and yielded 113,251 ounces of gold, or an average of 2 ounces 19 dwts. per ton. Nearly all this came from the upper workings.

per ton, but the contents of the tailings proved of much greater value than the amount of amalgam obtained. This led to a critical examination of the ore at the Sydney Mint. It was found that the bullion was of a fineness hitherto unknown in nature, assaying 99.7 per cent., occasionally 99.8, of pure gold, the rest being copper, with a trace of iron. It is remarkable as being almost entirely free from silver. Its value per ounce was £4 4*s.* 8*d.*, pure gold being worth at the London Mint £4 4*s.* 11½*d.* Dr. Liebius, of the Sidney Mint, considered, as the result of numerous experiments, that the iron present was in the form of an oxide, which coated the gold and so prevented its contact with the mercury.* Parcels of the ore were then sent to the Technological Museum at Melbourne, where Messrs. Newberry and Vautin subjected it to numerous tests. It was found that when dehydrated by heat it became less refractory; and its treatment by chlorine was suggested.† The ordinary Plattner process, as worked in California, was found quite unsuitable, owing to the extremely fine crushing which it required, and which resulted in the formation of slimes, greatly hindering subsequent filtration. It was then that Messrs. Newberry and Vautin planned the method which has since, with the addition of some mechanical details, been patented under their joint names.‡

While the stamp-mill proved incapable of extracting a proper percentage of the gold, the peculiar character of the ore is remarkably suited by the process now in use. The extremely minute state of subdivision of the gold and the very friable nature of the gangue caused, in wet stamp-milling, a considerable loss in "float," carried away in the readily-formed slimes; but the character of the ore indicated that dry crushing would prove effective; and hence the early introduction of Krom rolls was the first great improvement made. The old works, which are just above the stamp-mill, receive their supply

* My own experience with the ores of Gilpin county, Colorado, leads me to believe that this is, more frequently than is usually supposed, the obstacle to successful amalgamation.

† It had been previously treated with chlorine by Limberner at Gympie.

‡ The mine-manager informed me that nothing of the original Newberry-Vautin plant remained save a piece of old shafting. The Mount Morgan Company bought the right to use the process, and hence does not pay any royalty. While the original plant has been entirely replaced, it cannot fairly be said that the method of treatment has been changed, or that it is any other in its main features than that known by the names of those who first adopted it. Into the question of the originality of the process it is not within the province of this paper to enter. It has been ably discussed by the New York *Engineering and Mining Journal.*

on an aërial tramway, three-fourths of a mile long, with a total fall of 400 feet. It is of the Hallidie type, uncommon in the colonies, but familiar to American mining engineers, and need not therefore be here described. The feeding of the buckets (of which there are sixty) is admirably effected by the use of a travelling supply-bucket. The old works contain two rock-breakers, one revolving ore-dryer, four sets of rolls, ten furnaces, fourteen barrels, twenty-six vats and twenty-eight filters. The power is supplied by one 40-H. P. (nominal) engine and two 30-H. P. boilers. The practice in these works is identical with that in the new works about to be described.

The new works, erected on the higher slope of the mountain, about 400 feet from the summit, while somewhat similar to the lower works, show improvements in arrangement, besides allowing the necessary fall for the tailings. The ore, coming from the open cuts and the Freehold tunnel by a series of inclines (some of which are automatic gravity-tramways) passes through two Blake-Marsden rock-breakers, and is then dried in two revolving cylindrical furnaces, self-discharging after the manner of a *trommel*. The rotation is produced by an external cog-wheel. Thence the dried material goes to Krom steel rolls, arranged in two series of three each. Each set of rolls is followed by a revolving sieve, the coarse stuff passing into an elevator which returns it for second crushing, while the fine passes on to the second rolls and thence to the third. The last pair of rolls pulverizes the ore twice before it is carried by a worm-conveyor to an elevator which sends it to a chamber above the furnaces. An 80-H. P. compound engine drives the rolls and a 20-H. P. engine the breakers and revolving dryers, while two Root blowers are employed in the removal of the dust, of which there is a great deal. Five Cornish boilers of 30-H. P. each supply the necessary steam.

The furnace-chamber is 400 by 120 feet in size, and contains 18 ordinary reverberatory furnaces, each having a capacity of 1 ton. The heat to which the material is here exposed is a cherry-red, such as will get rid of organic matter, water of crystallization, etc., but does not approach that of a thorough calcination. At the end of 3 hours the charge is removed and spread upon the cooling-floors. When it has cooled it is trucked to the 21 iron hoppers which supply 21 chlorination-barrels. The charge for chlorination is 1 ton of pulverized ore, 90 gallons of water, 40 to 50 pounds of sulphuric acid and 35 pounds of chloride of lime. It will be seen from the nature of the charge that the chlorine is generated in the barrels

themselves, and attacks the gold while in a nascent state.* In order to make the charges uniform, the ore is mixed so as to yield 4 to 5 ounces per ton.†

The chlorination-barrels have been much altered from time to time. They are now made at the mine out of the native timber (*Eucalyptus*), the staves being made of "spotted gum" and the ends of "iron bark," with an inside diameter of 3 feet 6 inches. A cast-iron frame was at one time placed round the outside edges. The barrels are tarred to render them tight and are protected inside by a double lining, an innermost wooden lining protecting the lead one. They revolve on horizontal bearings, the power being supplied by a belt travelling around the barrel itself. Six revolutions per minute is the usual speed, and the charge is removed after an interval of from 1½ to 2 hours.

The barrels discharge into 84 leaching-tubs. These tubs or vats will hold 2½ to 3 tons of pulp each, and are supplied with filter-beds of sand and gravel, through which the solution passes before it is conducted by lead and antimony pipes to the charcoal-filters. These are 64 in number, V-shaped and packed with 2 feet of charcoal of unusual fineness, the upper layers having the size of coarse blasting-powder, and increasing in fineness toward the bottom. From 9 to 14 bushels are used in each filter.

When saturated with the precipitated gold the charcoal is removed to the assay-office, where it is burned to an ash containing 75 per cent. of metallic gold. The solution from the filters passes into tanks of concrete, whence it is pumped back to be used again in the chlorination-barrels.

The sulphuric acid used in the generation of the chlorine is manufactured upon the ground in leaden chambers, having a capacity of 20 tons per week. It is estimated that by the production of the acid on its own premises the company saves £20,000 per annum. There are also extensive work-shops, with lathes and shearing-machines, together with two assay-offices, well equipped with all the necessary appliances. The whole plant is run continuously, being lit by electricity at night, when the mountain assumes a very beautiful and animated appearance.

The mine and all its surroundings bear testimony to the great energy displayed in its development, in keeping with the value and size of the great ore-deposit which has made Mount Morgan famous.

* After the style of the processes patented by Mears in 1877 and De Lacy in 1864.

† The pleasure of producing a uniform mixture of such high grade has not been possible since the time of my visit.

The accompanying plate, made from a photograph, gives a good idea of the surroundings of the mine. In the foreground is the bridge crossing the Dee creek; immediately above is the old stamp-mill; behind it is a pile of the ore first broken; and behind that

again are the lower assay-offices. To the left are the old chlorination-works, while the upper chimney indicates the new works. Beyond is the Mount itself, the white spot being the iron roof of the main shoot which leads from the upper workings.

SAMPLING ORES WITHOUT USE OF MACHINERY.

BY WILLIAM GLENN, BALTIMORE, MD.

(Cleveland Meeting, June, 1891.)

THE taking of proper samples of crude ores seems to be less thoroughly understood, or less carefully practiced, than its importance requires. We all know how often we encounter the reports of very accurate assays and analyses, the weight of which as evidence wholly depends on the method of obtaining the sample, and is very probably *nil*, because there has been no proper sampling. Passages might be cited from technical books, and from current technical literature (all the work of writers whom we delight to honor) to show that even among high scientific authorities there is too little acquaintance with the practical art of the sampler. There is no metallurgical or chemical establishment which does not frequently receive samples truly representing nothing. They consist usually of bits of ore, or what not, selected because they are worse or better than the average of what they are meant to represent. And they are worthless, no matter who made the selection. Where conscious choice is permitted to enter into the operation, a fair sample will not result, unless by a miracle.

The work of sampling is often looked upon as within the realm of boys and of pensioners only. At least, though the manual labor be left to subordinates, the principles on which it is conducted and the safeguards with which it is surrounded are not unworthy of the study of experts; and experts should be ready to teach these principles and enforce them in practice. For exact sampling is the indispensable first step towards learning the value of any boxful, carload or shipload of ore.

I propose to give a simple account of the method of sampling by hand, such as I have always pursued. There is nothing new about it. So far as I know, it is the immemorial old method, as old at least as Cornish copper-mining. Perhaps my fellow-members in the Institute may suggest improvements in it. If not, it will constitute a record in our *Transactions* to which laymen and beginners may be referred.

There are two principal processes to be considered: first, how to take the gross sample of the lot of ore; second, how to proceed with that sample. Really, there is no iron-bound rule governing the first step; each may have a way of his own; yet most samplers proceed in much the same way. But once having the rough sample, there is no question as to what shall be done with it. It is to be worked down after the orthodox fashion. It must be broken and mixed and quartered until only a few ounces remain. The sampler bottles this residue; the chemist does the rest.

Assume that we have a rough sample, weighing a ton, of any ore with its gangue. Assume that by some manner of magic we can at once reduce the whole of it to dust. Evidently, if we could mix it long enough and well enough upon a clean, tight floor, it would finally become homogeneous. In that case, we could fill half a dozen small bottles from any part of the pile, and they would be fair samples. But the work would have to be admirably done; so well done that a single gramme taken from the pile would properly represent the entire ton of sample. Really, this would entail a great deal of labor. And if the rough sample weighed 15 tons or more, as it would if taken from a shipload, the bare labor of mixing that quantity of dust until homogeneous is wearying even to think of. We cannot proceed upon the proposition that a final sample may be obtained in any such way. Yet upon a rock similar to this many are wrecked.

To take, for illustration, a definite case, let us assume that we have to sample a 10-ton pile of 10 per cent. copper-ore, prepared for market. It will consist of masses generally the size of one's fist, together with masses of all smaller sizes and even of dust. We shall require for the work a clean, tight floor or pavement, an iron mortar and pestle, a shovel, a small hammer, a piece of iron for an anvil and, lastly, a broom. Besides these, we shall find convenient a wheelbarrow or a barrel or box of some kind.

For convenience and force I will put my description into the form of homely directions, such as I might give to my workman.

Begin by shovelling the pile roughly into the form of a flattened cone or a flattened pyramid; say we choose the pyramid. Now make a trench straight through the pile, cutting it into two nearly equal parts. And again by a trench (at right angles to the first) divide these halves into four nearly equal quarters. A part of the ore taken from these trenches will form the sample required. Proceed as follows:

Having the wheelbarrow ready, begin at the middle of any side

of the made-up pile and cut the first trench. Cast the first shovelful to the right, the second to the left, the third into the wheelbarrow. Repeat this order of shovelling until the barrow is full, then empty it upon the well-swept floor intended to receive the sample. Continue in the same way until the trench has passed through the pile, when there will result two rather long and narrow piles. Begin the second trench, extending it across the middle of the two piles, casting the first shovelful right, the second left, the third into the barrow. Proceed in the same way as with the first trench. When done, you will have shovelled about 6000 pounds of ore. As every third shovelful was thrown into the barrow, there will result about 2000 pounds of sample upon the floor. That this is a fair sample of the original pile is based upon the assumption that each third shovelful thrown into the barrow was like the first and second ones cast into the piles. The hypothesis is reasonable and freely to be trusted.

Having the sample, proceed with it after the regulation method, as follows:

Spread it thinly upon the floor; now examine it. If there be any lumps which look larger than the general run, place the anvil upon the pile, and between that and the hammer break those lumps. The next step is to thoroughly well mix the sample. Begin at one edge of it and shovel the ore over upon itself. Move around to the opposite side of the pile, and from that side shovel the ore again upon itself and back to its original place upon the floor. Having it well mixed, form it into a flattened cone and sweep all the dust upon and around the pile. You have now to halve and quarter the sample as follows:

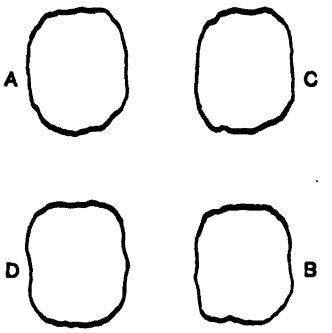
Commence at any point and shovel a road through the center of the pile, casting the shovelfuls alternately right and left as you proceed. This movement will result in cutting the pile into two elongated nearly equal ones. Beginning at the middle of one of them, shovel a road through it in the same way as was done before. And in precisely the same way cut the other pile in two; sweep upon each pile the dust belonging to it. These movements will result in four piles, as in Fig. 1.

If the sample were well mixed, when and as directed in these notes, then will each of the quarters, A, B, C, D, have the same composition as have all the others. But if, upon inspecting them, you judge one or another to be poorer or richer than the others, you will have then sufficient evidence that the work has been badly performed. In that condition of affairs, mix well together all the piles, and once more

halve and quarter them. Having made all the quarters of the same composition, it follows that any two of them may safely be accepted as representing the original 2000 pounds of rough sample. This opens a road leading in the right direction, since it enables us finally to get rid of half the sample. We may cast out two of the quarters and retain the other two for the sample. It is a matter of indifference which two are retained, say A and B. Remove from the floor C and D, together with the dust belonging to them.

We have again to break the larger stones, until there remain none larger than walnuts. Place the anvil between the piles, within easy reach of them. Take a stone from A, break it; take one from B,

Fig. 1.



Plan of ore-piles after quartering.

break that. Continue in this way, taking stones alternately from each pile, until all are reduced to the size stated. By proceeding in this way, the sample is more or less mixed while being broken. Complete the mixing as before, by shovelling all of the sample forth and back over the floor. Form it once more into a flattened cone, and sweep the dust upon and around it. Divide the cone into two halves, and those into four quarters, precisely as you before did, and as illustrated in Fig. 1. You have now to reject two of these quarters. The unwritten law of the sampler says that it must be those holding the positions A and B, because those were retained in the last quartering. Remove A and B from the floor, retaining C and D for the sample. These would now weigh about 500 pounds.

Proceeding as before, break down the lumps of ore until none are left larger than say 1-inch cubes. Again mix well the sample, make it into a pile, sweep up the dust, halve and quarter the pile. Reject two quarters (C and D, of course), retain two, precisely as in former quarterings.

Once more break the lumps, this time down to half-inch cubes. Mix well the sample, make it into a pile, sweep up the dust, halve and quarter, reject two quarters. The two quarters retained would weigh about 125 pounds. Break it down until comparable to fine gravel and coarse sand. Mix and quarter once more.

The two quarters this time retained would weigh about 60 pounds. With the mortar and pestle break this to something approaching coarse sand. Again mix and quarter. The quarters this time retained are to be ground yet finer, mixed and quartered.

If you have no mortar and pestle, the hammer and anvil may be substituted throughout. After getting the material into the form of coarse sand, it is best to mix and quarter it upon a sheet of paper, even an old newspaper.

At this point the sample would weigh about 15 pounds; its larger grains would be in size like coarse sand. It would be safe now, without further breaking, to mix and quarter it twice, or until its weight did not exceed 4 pounds. Run this through the mortar and then mix and quarter it twice, or down to 1 pound weight. Grind this to something approaching powder, and, for the last time, mix and quarter it. Have ready 6 wide-mouth 1-ounce bottles. Place them in a line, side by side, upon a sheet of paper. From the other paper pour the ground sample in a small stream, forth and back across the mouths of the bottles, until they are all full up to their shoulders. Cork, seal and label them, and the sampling is done, and properly done.

If there should lurk in your mind a suspicion that this half-pound residue of dust may not, after all, properly represent the rough sample with which you began, go back over the work, and try to decide precisely where in the quarterings the sample retained ceased to be a sample. If you can decide upon that point exactly, then you will know just where you failed to do your work properly. The error will be with you and not in the method.

It does not, in the slightest, matter of what solid a sample may consist, or how much or how little it may be, it should be worked down in the manner just detailed. Whether a sample consist of 20 tons or of 1 ounce, it matters not, except as to breaking and grinding.

A word may be added as to larger and rougher ore-piles than have as yet been mentioned. It is not unusual for one to have a pile of 100 tons, or 200 tons, which one would like to sample. Such piles are apt to consist of lumps larger than a man's head, together with masses of all smaller sizes. Where a pile is formed by dumping ore

uniformly upon its top, the likelihood is that the pile is homogeneous. In such a case, it is safe to make short cuts into it at several points around its base, and to consider as sample the ore so gotten. It is safer to make one cut through the pile, retaining as sample each third shovelful, as in the case of the copper-ore we have just considered. In forming ore-piles of the weights given, it is a good custom to put upon a separate platform each tenth or twentieth barrow-load coming from the mine; the small pile will prove a fairly good sample of the large one. But no matter how it may be gotten, the rough sample is to be broken and mixed and proceeded with after the regulation method.

No account is here taken of moisture-samples, or of sampling train-loads or ship-loads. This paper is meant to help beginners, but it would be, in the writer's judgment, a benefit to others also, if our members who are practically acquainted with sampling would present their own methods with regard to the various branches of the art. The precautions specially required in sampling rich silver-ores, for instance, might well be made the subject of comparison and discussion.

Discussion.

R. W. Raymond, New York City: I am glad that this subject has been brought to the attention of the Institute. Mr. Glenn has by no means over-stated its importance; and yet, with the exception of the paper of Mr. D. W. Brunton (*Trans.*, xiii., 639) describing a mechanical ore-sampler, I can recall little or nothing in our *Transactions* concerning it. The practice in the West, with regard to silver-ores, etc., is affected by various considerations. As a general rule, such ores are either already fine or are crushed before treatment. I think it is now generally admitted that a good mechanical sampler connected with the crushing-machinery takes a fairer average sample (usually one-fifth or one-tenth of the total lot of ore, according to the richness and the consequent importance of exact determination of value as a basis of payment to the miner) than can be got by throwing aside the fifth or tenth shovelful in unloading. At all events, the mechanical sampler cannot be suspected of bias, conscious or unconscious, provided it is so constructed as to deliver impartially the coarser and finer particles of the stream of ore passing through it. Disputes are therefore avoided by using it.

But for many classes of ore, the sample is still taken with the

shovel; and it is after this has been done, whether mechanically or by hand, that the process begins, which can properly be compared with that described by Mr. Glenn. The differences between this process and his are required by the following principal considerations:

1. The material, namely, ores carrying, besides lead or copper, more or less silver and gold, is so much more valuable that small errors in determining its contents may be the source of large loss to the buyer or seller.

2. It is almost always finer than the crude material contemplated by Mr. Glenn. Either it has been crushed before the sampling begins, or it is so generally fine when received that the preliminary stages of sampling may be, in the judgment of the manager, performed without crushing. This, however, always involves a risk, the assumption of which by a prudent manager is justified only when the conditions are favorable; for instance, when the character of the ore is well-known; when its grade is not too high; when the difference in richness between coarser and finer pieces is not great; when careful experiment has shown that sampling without a preliminary crushing is sufficiently accurate; when avoiding the extra cost of crushing would be an important saving, or when the crushing machinery of the establishment is fully occupied with material which must be reduced before sampling. The nature of the contract between the smelter and the miner is another element to be taken into account. Perhaps the ore is not to be paid for strictly according to sample-assay (a thing which now and then happens, though not often, in the purchase of large dumps at a round price, or the treatment of ores on joint account, to be settled by the net results, instead of the preliminary assays, or under special contracts, in which the loan of money or other inducement to the miner has secured special terms). Perhaps, again, the smelter owns the mine, and can thus afford to save current expense by taking some risks in sampling his own ore which he would not take with ore for which he had to pay according to the sample-assay.

I think it may be said, however, that the sampling-works proper, that is, those which are not parts of smelting-works, but stand between the smelter and the miner, as agents for the latter in the sale of ore, warranting to the purchaser the accuracy of their determination of its contents, always crush everything they receive. And I think it may be said, also, that when the capacity of crushing and sampling-apparatus is adequate, and the arrangements for receiving and hand-

ling ore are so perfect as to avoid both demurrage on railway-cars and re-handling of material, the crushing and mechanical sampling of all lots received is not only better, but cheaper, than shovel-sampling of crude cargoes. The matter may indeed be left to the discretion of a competent manager; but the most competent managers are not anxious to multiply the matters which are left even to their own discretion, still less to the discretion of subordinates. There are innumerable things which have to be watched daily and hourly around great smelting-works; and it is a relief when any one department can be placed in the category of comparative routine, calling for fidelity rather than discretion. Moreover, the exact determination of all the elements of an ore of the class here contemplated is as important on technical as on commercial grounds. Without it, the metallurgist can neither calculate his charges successfully nor be held responsible if he fails to do so. And the sample taken of each new lot of ore received is not only assayed for its valuable contents, such as lead, copper, silver and gold, and for sulphur, iron, silica and zinc (which may affect the price to be paid for it), but also analyzed for all its earthy bases, so that it may be properly mixed for smelting with other ores or fluxes. Now it is quite likely that the gangue-minerals of an ore may break very differently, according to their natural hardness or cleavage, in the ordinary processes of mining and shipment. Moreover, the same car-load from a given mine may contain ores from different stopes, varying so greatly in character and condition that even if the average of gold or silver were reasonably uniform, the average of silica, lime, magnesia, baryta, etc., would not be so, and the lumps might be entirely different in these particulars from the finer stuff.

With these digressions and explanations, I repeat the general statement that the ore to be sampled in the West is generally fine. But for the rough sampling of ore at the mine, or in localities where proper crushing cannot be done, the procedure described by Mr. Glenn seems to be a good one for the preliminary stages.

3. Notwithstanding the general fineness of the crude pile, and no matter to what degree it may have been reduced by crushing (short of the final reduction to impalpable powder), there is still a distinction between coarser and finer particles, and in the majority of cases this distinction is one of value as well as size. Many of our western ores carry silver, for instance, not only as an accessory ingredient of galena, but also in the form of finely-disseminated "true silver-ores" accompanying the galena. These are usually not only in fine crys-

tals or specks to begin with (often not distinguishable to the naked eye), but brittle besides, so that the shocks and abrasion of mining, transportation, crushing and handling reduce them to still smaller size. In the stamp-mill they make "slimes." In concentration they are very likely to escape and leave the headings impoverished, though clean from gangue—"poor but honest," so to speak. In sampling, they present the greatest danger of error.

Something similar might be said concerning the tenor of gold in many ores. Although true gold-ores are not common, yet in a lot of ore consisting largely of auriferous pyrites, for instance, there will certainly be more or less oxidized material, in which there is fine, free gold. I confess, however, that so far as my observation goes, this is not the source of as serious difficulty in sampling as are the brittle ores of silver. That is a point on which I hope some of our members who have had wider practical experience with the sampling of gold-bearing ores will throw further light. My own knowledge comprises more particularly the practice of lead-smelters; and in that practice the gold in the ores purchased, though it is determined and paid for, has not been hitherto a point raising special questions of difficulty. Or rather, the precautions taken as to the fine dust of silver-ores cover the question of gold also, so that this has not called for separate consideration. Moreover, almost all the distinctively gold-bearing ore of the country, as well as a great part of the silver-ore, is reduced by wet-crushing for amalgamation or other subsequent treatment, and consequently sampled by battery-assays—a process which is not here under consideration, and which does not encounter the precise difficulty just described. I am inclined to think that in lots of crude fine ore, or of ore dry-crushed for sampling, oxidized material containing gold would not present as much difficulty as silver-ore, because it is less likely to form fine flying or sifting or rolling dust. The iron oxide tends rather to pack or adhere—at least to lie where it falls—unless it has been artificially dried and pulverized.

But the difference of value between coarse and fine particles may be due to other minerals than those already mentioned. And there may even be cases in which the greater value is in the coarser material. I need not, however, enlarge further under this head.

4. Obviously such a difference would have no effect, if the whole mass to be sampled were so thoroughly mixed that the proportions of different sizes were uniform throughout. But this is not practicable, because the coarse and fine particles behave differently as to

rolling and sifting over or through the pile upon which they are shoveled. Consequently, it becomes necessary, in order to secure a correct final average, to make the distribution of sizes in each pile formed *symmetrical,* since it cannot be made truly *uniform.* At a later stage, as will be seen, the whole of the sample thus secured is reduced to powder. Yet still the precaution of making symmetrical piles for quartering is continued to the last.

With these explanations I will describe the quartering of a sample of ore (say one-fifth or one-tenth of a gross lot) as practiced at some of the leading works in the West.

The mass is first shoveled into a ring on the sampling-floor, and this ring is then shoveled toward the center, each shovelful being carefully delivered upon the summit of the pile in the center, so that it shall roll equally in all directions. A conical heap having thus been formed, it is pulled down and spread out. The workmen walk round and round the pile, pulling with the shovel, as it were, the ore towards them, so that it rolls outward. The lower six inches of the pile is not disturbed, and when this process is finished, the conical heap has become a truncated cone of larger base-area and six-inches height. This flat heap is now quartered by pressing a stick or a board held edgewise down into it so as to mark the diametrical divisions. Two opposite quarters are cut out with the shovel and removed. The other two are again mixed, formed into a conical heap, and flattened out as before. This procedure is repeated until the quantity has been reduced to one or two wheelbarrow-loads, when, if the material has never been mechanically crushed, it is crushed in the rolls to say half-inch maximum size. The quartering is then continued till the sample has been reduced to a panful. This is ground say to 60-mesh size (after a partial preliminary drying, if necessary to facilitate the grinding in a rotary fine-crusher) and then taken to the assay-laboratory, where it is thoroughly dried (say for twenty-fours at 212° F.), and rubbed fine until the whole will pass through an 80-mesh sieve. Quartering is then resumed and continued until the sample is only sufficient to fill three bottles, one of which is for the assay of the works, one for the customer, and the third for the umpire-assay, if such should be required.

The details of this practice may vary in different works, and I trust our members, many of whom are engaged in such work, will give us, in the way of criticism or suggestion, the benefit of their experience. It is perfectly evident, as Mr. Glenn says, that a vast amount of skill and precision is daily wasted by our chemists in the

delicate analysis of samples that mean nothing. To cite a single instance, I have recently studied with much interest the various papers in our *Transactions* concerning the practical results of the magnetic concentration of iron-ores. In Mr. Birkinbine's paper, presented at the New York meeting, there is a simple formula, by which he calculates from the percentages of iron in the crude ore, the concentrates, and the tailings the quantity of crude ore required to furnish a ton of concentrates; and he applies this formula in testing the comparative economy of different magnetic concentrators. But in several instances the actual quantities of crude ore, concentrates, and tailings have been reported, and they do not agree with the results of calculation by this formula. Now the formula is mathematically accurate, and there is no reason to doubt the accuracy of the chemical work. It follows that the samples, either of the crude ore, of the concentrates, or of the tailings, could not have been accurate. It is certainly to be desired that future tests of such work should include more thorough sampling, and that future accounts of such tests should include descriptions of the method of sampling employed. In the absence of certainty as to the fundamental data, the application of mathematics to the discussion of results seems to be labor thrown away.

STONE-COAL IN THE LEAD BLAST-FURNACE.

BY JAMES W. NEILL, E.M., HARRISON REDUCTION WORKS, LEADVILLE, COLO.

(Cleveland Meeting, June, 1891.)

BITUMINOUS coal has for many years been used for the smelting of iron-ores in the blast-furnace. In some districts in Scotland it is used alone, in others it is used mixed with coke. The similar use of certain bituminous coals in the United States has been repeatedly mentioned in the *Transactions*. (See, for instance, as to the Hocking Valley coal, the papers of Dr. T. Sterry Hunt, *Trans.*, ii., 275, and vii., 313; and as to Indiana "block-coal," the paper of Mr. J. S. Alexander, *Trans.*, i., 225.)

Phillips (*Elements of Metallurgy*, p. 250) says: "Raw coal for blast-furnace use must be of a non-coking kind, and, when of suitable quality, is perhaps the most advantageous fuel." In the first

volume of the *Chemical Technology* of Groves & Thorp, an article on "Fuel and its Applications," by Mills and Rowan, presents an exhaustive comparison of the values of coal and coke in iron-smelting, from which I quote the following:

"There is a still further circumstance in favor of using raw coal in smelting iron. We have seen that, whether coked or raw, in a heat-giving point of view, there is not much to choose between the two: that while 4 cwts. of coal per ton of iron are wasted by inferior oxidation in the furnace, the same quantity is lost in the coke-oven. In the former case, however, we have, in addition to the inflammable carbonic oxide of the escape-gases, about 7 cwts. of combustible gases which are available for other purposes; whereas, in coke-ovens but a very small percentage of these remains over after satisfying the requirements of the process of coking itself."

In the lead-silver-smelting blast-furnace, the requirements of iron-smelting are not present. Here the general question is, Which fuel, or fuel-mixture, will permit the most rapid driving of the furnace? The conditions of efficiency in the reduction of iron, of sufficient heat, of capacity to carry the burden, etc., are usually satisfied by any of the commercial cokes of the regions surrounding the lead-silver smelting-districts, and it is therefore usually the price which decides the choice of coke. In most of these smelting-districts, bituminous coal of non-coking character is very much cheaper than the coke, and its use alone or with coke would materially lessen the fuel-expense per ton of ore. This saving, if achieved without occasioning other losses in the working of the furnaces, would be net gain to the silver-lead smelter, and the following experience with the use of bituminous fuel is given to show what can be done in this way.

About 1884, while in charge of the smelting-works at Mine La Motte, Mo., I ran short of coke, and having a supply of stone-coal on hand, I replaced half of the coke in the charge with this coal, continuing its use until a supply of coke arrived. During this period (about twenty-four hours) no noticeable change in the working of the furnace occurred; but as the stone-coal was more expensive than the coke, the practice was not continued. With the precedent of the authorities I have cited,* and my own brief personal experience, and in view of the circumstance that in Leadville to-day coke costs three times as much as certain kinds of coal, I have recently ventured to

* To these may be added the paper of Mr. A. Eilers on "The Metallurgical Value of the Lignites of the Far West," *Trans.*, i., 216, in which unsuccessful experiments are alluded to, but pronounced to be, in the author's judgment, inconclusive.

experiment with Rocky Mountain coal in the blast-furnaces of the Harrison Reduction Works.

These furnaces are 78 by 36 inches in size at the tuyere-line, have 10 inches bosh in the jackets, and are about 12 feet high from tuyere to charge-door. At the time the experiments commenced, the furnaces, having been running some time, were in bad condition from zinc-accretions in the upper part of the stacks, and would have to be "blown down" and "barred out" in a few days at farthest; and I reflected that, if the coal should prove impracticable as fuel, this event would only be slightly hastened.

On January 20, 1891, I replaced on No. 1 furnace 50 pounds of Cardiff coke with 60 pounds of lump-coal of a non-coking variety. The fuel-charge before the change had been: coke, 185 pounds; charcoal, 65 pounds. It was now: coke, 135 pounds; stone-coal, 60 pounds; and charcoal, 65 pounds. The charge was made on the evening of the 20th. Next morning showed no appreciable change; slag-assays were good, but in the afternoon the slag commenced to thicken and get colder, and finally refused to run out of the tap-hole, filled all the tuyeres, and compelled a stoppage of the furnace. On taking out the tap-jacket, I found that the charge had slipped down, filling the basin with raw material, which had stopped the slag. We heaved out a quantity of this raw material and cleaned the tuyeres, and put on the blast again, when the furnace cleared itself without further serious trouble. Much to our surprise, we found the top of the furnace in better condition, the amount of accretions hanging on the sides being much less and the charges settling more evenly.

On the 22d, as No. 1 continued to do well on this fuel, I put the same fuel-charge upon furnace No. 4, the upper part of which was also in bad condition. Here the result was the same; but by careful watching serious trouble was avoided. Meanwhile, instead of our having to clean No. 1 in a day or two, it ran thirteen days after the coal was first put on.

During the first week in February, all the furnaces were blown down and barred out, and blown in again on the above fuel-charge, all starting off nicely. On the 4th of February the charge was changed to: coke, 120 pounds; stone coal, 70 pounds; charcoal, 70 pounds, and on this charge they ran until the 14th. During this period the coal used was a lump-coal from Rouse, Colorado, a semi-coking coal of good quality, a sample giving us 8.91 per cent. of ash. The pecuniary saving was that, in fourteen days,

We replaced 126.98 tons of coke at $8, costing . . .	$1015 84
With 148.15 tons lump-coal at $5.50, costing . . .	814 82
A difference of	$201 02

On February 14th I replaced the lump-coal with an equal weight of pea-coal from the Sunshine mines. A sample of this gave us 7.72 per cent. of ash; it is a semi-coking coal, and costs us $2.50 per ton. The furnaces ran, for the remaining fourteen days, on the same charge of fuel as during the first half of the month, doing their work nicely.

During this time we replaced 126.84 tons of coke at $8, .	$1014 72
With 147.98 tons of pea-coal at $2.50, ...	369 95
Saving, ...	$644 77
Total saving for the month of February, . . .	$845 79

Which amounts to about 26 cents per ton of material smelted.

During March we used about the same fuel-charge, the coal being part nut, part pea and part lump, each used separately; and the total saving by the use of these fuels in place of coke was, for March, $1075.08, or about 30 cents per ton of material. For a few days I increased the amount of stone-coal to 100 pounds, using with it 100 pounds of coke and 70 pounds of charcoal; but the furnace did not work well on this charge, the hearth becoming clogged, and slag frequently flooding the tuyeres. Whether this was due to the fuel or to bad charging remains to be proved by further tests.

Aside from the direct saving by the use of the stone-coal, we have observed the following advantages in the working of the furnaces:

Below.—The slags appear better reduced and hotter, and the matte separates very well. Slag-assays have been, if anything, lower than on the old fuel; and this is particularly the case when the charges are hanging above. The jackets keep hotter and the tuyeres brighter; thus the fuel is more completely consumed; the furnace "rods" more easily; the crucible keeps open better; the lead is hotter.

Above.—The volume of smoke is somewhat increased and smells decidedly "tarry," but does not look different from that of the old fuel; the charges settle much more evenly and the fire does not creep to the top; thus the furnaces seldom flame. As the tops keep much cooler, the production of flue-dust is much smaller, and the losses of metals by volatilization must also be diminished. The furnaces now

run about a week longer than formerly before needing to be barred out; and this operation is no more difficult than before; but on blowing them down, the charge now sinks much further before flaming commences.

The pressure of the blast has not changed materially from that of the old fuel-charge, but now remains more uniform. The necessary reduction of iron can thus always be relied on.

To the practical furnace-man, these advantages are of almost as much importance as the direct saving in the cost of smelting.

The short trial with a higher percentage of coal I do not consider conclusive, and I am now working the amount of coal up by gradual steps to see what proportion we can use to the best advantage. The limit will probably be reached sooner on our low furnaces than it would be on higher ones. We are now using 27 per cent. of coal on the fuel-charge, and I believe that on our furnaces the limit will be reached at about 33 per cent., but that on higher stacks the percentage could be carried to at least 50.

Of course, this will in a measure depend on the quality of the coal used. For instance, I was obliged recently to use coke and coal from southern Colorado (containing 20 and 13 per cent. of ash respectively) in place of our regular fuel (containing 11 and 8 per cent. respectively), and although I made allowance for the difference in fuel-value and added flux for the ash, we had much trouble with the furnaces, owing to scaffolding of charges and incomplete combustion before the tuyeres, with consequent bad effects on the slags, etc.

There are many smelting-districts where the difference between the prices of coke and coal is even greater than here in Leadville. In such places the direct gain by the use of coal instead of coke will be correspondingly greater. But to practical furnace-men the other advantages above described will be, perhaps, sufficient to induce a trial of this simple experiment, even where the item of saving in fuel-cost is not important; and I shall be glad to hear of the experience of others in this line.

Discussion.

Arthur S. Dwight, Pueblo, Colo. (communication to the Secretary): Mr. Neill's experiments with bituminous coal in lead-smelting open up many interesting possibilities regarding the line to be followed in the future development of this branch of metallurgy. In Colorado, particularly, the large number of smelting-works erected within the past few years has caused the keenest competition, reduc-

ing smelting-margins on ores of all characters to figures which will barely cover cost. Indeed, on certain kinds of ores, there is a loss of two or three dollars per ton. Under the pressure of these conditions, lead-smelters have been forced to adopt every plan which gave any promise of reducing the cost of treatment, whether by cleaner and better metallurgical work, by saving in labor, or by cutting down general expenses per ton, through the building of additional stacks and the treatment of a greater total tonnage. The last-named means for reducing cost of treatment has proved somewhat disappointing. It is, indeed, in vain to expect a profit from smelting a greater tonnage of ore, on which there is a loss, unless that loss be well within the item of general expense, as assessed on the smaller tonnage.

Mr. Neill's saving of twenty-five to thirty cents per ton in the cost of smelting by substituting coal for a portion of the coke is, therefore, most important, and in many cases would change loss into profit.

From a technical standpoint, his experiments seem to have been remarkably successful, though it is to be wished that he had given us more complete details, for instance, as to the manner of charging the mixed fuels with the ore-charge; more definite figures as to the effect on the slag and matte; and a precise comparison of the speed of the furnace on coke and on coke mixed with stone-coal. The good behavior of his furnaces, which Mr. Neill chronicles, is evidence that there may be some decided advantages in the coal even where the commercial saving would not be so great as in Leadville. But it certainly seems like pushing things to an extreme to make a shaft-furnace 12 feet high manufacture its own coke, and have that coke ready to burn before the tuyeres in less than four hours' time.

To the writer it seems that there is a better field for cheapening the cost of smelting by doing without charcoal, which is more than twice as expensive as coke, and is now used solely on account of the greater tonnage it will allow to the smelter as compared with coke alone. At the works of the Colorado Smelting Company, at Pueblo, Colo., the proportion of fuel used is 128 pounds coke and 35 pounds charcoal to 1000 pounds of ore-charge (slag being neglected). We hope to have an opportunity to verify Mr. Neill's results under our conditions, but thus far we have not been able to make the trial.

In this connection it may be interesting to note the results of an experiment to replace coke with anthracite in a lead blast-furnace.

The credit for this experiment is due to Mr. Alfred Ropp, at present connected with the Selby Smelting and Lead Company of San Francisco, and the details are published with his permission.

In 1887, while Mr. Ropp was in charge of the Tomichi Valley smelter, at Gunnison, Colo., his supply of Crested Butte coke got very low. Some selected anthracite, of about goose-egg size, from the neighboring mines of Crested Butte, was on hand in the works, and as coke was very dear, he decided to try replacing some of the coke on his charges with this anthracite. It is important to note that *no charcoal* was used. Ten per cent. of the coke was first replaced with anthracite. The smelting-capacity of the furnace was slightly reduced; the slag, however, remained clean, *i.e.*, it carried 0.4 to 0.8 per cent. lead, and was low in silver. The percentage of anthracite was gradually increased, until 60 per cent. of the original charge of Crested Butte coke had been replaced with anthracite. The smelting-capacity of the furnace diminished as the proportion of anthracite increased. The slag remained about the same in lead and silver; likewise the matte, the latter containing about 8 per cent. lead and 4 per cent. copper.

During the whole of the experiment, the speiss was of the coarsely crystalline variety, showing that plenty of iron was reduced. The top of the charges remained cool and the crucible free. The tuyeres seemed to darken and get somewhat harder as the percentage of anthracite was increased. There was about 7½ per cent. of zinc in the charge during the experiment, but it seemed that less zinc-accretion was formed on the walls than when coke alone was used. As to the manner of feeding, the charges of ore and fluxes and the fuel-charges were distributed in even layers over the whole section of the furnace.

The experiment lasted 12 days; and at length, when 60 per cent. of anthracite was used, the capacity of the furnace was decreased one-third.

Under the conditions prevailing in Gunnison, particularly on account of the high cost of labor, the experiment was not a commercial success. The furnaces employed were water-jacket furnaces 80 inches by 36 inches in section at the tuyere-level, and 9 feet from tuyeres to feed-floor. Had they been higher, it is very likely that the results might have been better. Another condition that made the furnaces run more slowly was the extreme fineness of the charges, which became more and more effectively troublesome as the coke was cut down.

In both of these experiments, one with bituminous coal and one with anthracite, there are several advantages that command notice: 1, cool tops; 2, open crucibles; 3, good reduction, with clean slags and mattes low in lead; 4, less zinc-accretions.

The fact that Mr. Ropp used no charcoal, and worked with a very low furnace and a charge less coarse than the average, should be borne in mind in comparing the results of the two experiments.

The common points of advantage demonstrated in both cases are of sufficient importance to justify very extended experiments by lead-smelters with the use of coal in their shaft-furnaces.

SOME ONTARIO MAGNETITES.

Discussion of the Paper presented by T. D. Ledyard, Toronto, Ont., at the New York Meeting.

(Cleveland Meeting, June, 1891.)

R. W. RAYMOND, New York City: In his paper at the New York meeting (*Trans.*, xix., 28), Mr. Ledyard described, among other mines, the Belmont, in the county of Peterboro, Ontario, the developments of which at that time indicated, though they did not, perhaps, completely demonstrate, a deposit of high-grade Bessemer magnetite of very great extent. Mr. Ledyard said that experts estimated the quantity of ore within 100 feet of the surface at 1,000,000 tons. But the precise data upon which this estimate was based were not furnished. We are all interested in the development of fresh supplies of Bessemer ores. I think nobody who knows how enormous is the demand can seriously fear that such deposits will be exploited too rapidly. On the other hand, I need not here enlarge on the fact that a considerable number of enterprises, in which American ironmasters have been more or less interested, for the development of Canadian iron-ore deposits, have been unsuccessful, more by reason of disappointment as to the quantity of the ore and the facilities for cheaply mining it on a large scale, than as to its quality. Of course, a Canadian mine supplying the American market is handicapped at present, to the extent of 75 cents per ton, by the duty on foreign iron-ore; but this would certainly not be prohibitory in the case of a deposit sufficiently large, rich, and accessible. Or, to put it otherwise, there might easily be, somewhere in Canada, near our borders, a mine so favored that its product could be profitably

marketed on our side of the Lakes in spite of the duty. That this has not been done to any striking extent hitherto is negative testimony, tending to show that the best that Canada could offer was not economically good enough—or, to put it more precisely, not 75 cents per ton better than the average conditions of American mines actively operated in supplying the same market.

Now the analyses of the Belmont ore, furnished in Mr. Ledyard's New York paper, and showing an average of over 66 per cent. of iron with less than 0.02 of phosphorus, leave nothing to be desired on the score of quality; and the general description which he gives of the nature and position of the deposit, though lacking in details, seems to justify his statement that the occurrence resembles some of those in the Lake Superior ranges of magnetic deposits. As I understand from that paper, and from a sketch of the workings made at about the date of its publication, a considerable area of iron-ore, lying very near the surface, had been exposed by shallow pits and cuts. This area was perhaps 250 feet square; but there was strong needle-attraction, indicating a size of at least 400 by 600 feet. An average of 50 feet thickness over this area would give something more than the 1,000,000 tons estimated by experts, and it seems probable that some such calculation was the basis of that estimate. Mr. Ledyard is now present, and has brought with him for exhibition some cores from recent borings with the diamond drill on this property. I hope he will give us such information as he can concerning its most recent developments.

MR. LEDYARD: The developments made since the date of my paper show the ore-deposit to be much larger than at first supposed. The diamond drill-hole from which these cores were taken was bored vertically, near Pit No. 1, to the depth of 100 feet, of which about 75 feet appears to be ore—and mostly rich, clean ore, very free from sulphur, but carrying some lime. The following analyses have been made lately:

Pit.	Metallic iron.	Phosphorus.	Sulphur.	Silica.
A. No. 1,	55.240	.019	traces.	11.200
B. "	63.131	.023	.004	.....
C. No. 2,	60.376	.037	.007	.....
D. "	68.33	.016	.0375	.....
E. No. 3,	69.85	.013	.012	.....
F. "	70.326	.0056	.0023	.875

Analyses A., B., C., and F. were made by William Molin, 18 Liberty Street, New York. Analyses D. and E. were made in Milwaukee, on behalf of intending purchasers, I think, by Prof. Davenport Fisher, but I am not quite sure of this.

The Belmont Bessemer Ore Company (Limited), which has been organized to work this mine, has erected a boarding-house and office and is now testing further with the diamond drill. It is intended to connect the mine with the Canadian Pacific Railway at Blairton Station.

NOTES ON SOME OF THE MAGNETITES OF SOUTHWESTERN VIRGINIA AND THE CONTIGUOUS TERRITORY OF NORTH CAROLINA.

BY H. B. C. NITZE, BALTIMORE, MD.,

(Cleveland Meeting, June, 1891).

A DESCRIPTION of some of the magnetic ore-deposits in this region should be of interest to the mining and metallurgical public, inasmuch as very little has been said or written concerning them.

I refer to the magnetite deposits of Franklin and Henry counties, Va., and of Stokes county, N. C.

These deposits remained comparatively unexplored until recently, when the construction of the Roanoke & Southern R. R. through this territory has called attention to them, by furnishing the means of transportation for the ores. It is true that the ores have long been known, at least locally, and were smelted in catalan forges and old-fashioned, cold-blast charcoal-furnaces long before and during the late war. At the present time, however, the old workings are caved in, filled up and over-grown to such an extent that, as a rule, but little can be learned from them. No prospecting of any consequence has been done, and as my time was too limited and the territory to be examined too extensive to permit additional openings of a more satisfactory nature to be made, the incompleteness of some of the data I have to offer (notably, those referring to the absolute width of the veins) will be pardoned.

It will hardly be necessary to say that this ore-bearing formation belongs to the Archæan, or Eozoic age, the usually occurring rocks being schist, shale, gneiss, granite, sandstone and quartzite. The stratigraphy is, as a rule, quite regular (excepting in the Danbury section, Stokes county, N. C.), the general strike being northeast and southwest, and the dip nearly vertical. The country is gently roll-

ing and hilly, traversed by long ridges of moderate height, trending northeast and southwest, parallel to the Blue Ridge, and enclosing fertile and well-watered valleys. The soil is, in general, a gray, sandy loam, naturally adapted to the cultivation of tobacco, for which product this section enjoys a wide reputation.

I will divide the country to be described as follows:

I. Rocky Mount Section.
II. Brown Hill Section.
III. Stewart's Knob Section.
IV. Davis Mill Section.
V. Martinsville Section.
VI. Danbury Section.

I. The Rocky Mount Section.

Just outside and to the west of Rocky Mount, the county seat of Franklin county, are situated the Rocky Mount, or Clark mines, on a vein of magnetite which was worked here many years ago by the Pennsylvania Steel Company, and abandoned, presumably, on account of the cost of transportation.

The deposit exists in a long, low ridge, parallel to the "Grassy Hills." The outcrop is very distinct and has been stripped to a great extent by the Pennsylvania Steel Company. There are also a number of old shafts on the vein, which were unfortunately caved in and filled with water, so that I was unable to make an inside examination of the mines.

The last shaft was put down by the Virginia Development Company, to a depth of about seventy-five feet, but this was also filled with water. I am given to understand that the vein pinched out entirely at this depth.

From the investigations that I made I am satisfied that the character of this vein, as is the case with all magnetic veins in this section of Virginia and North Carolina, is lenticular, pinching and bulging out in all directions.

The general strike of the vein is northeast and southwest, and the dip from 60° to 80° southeast. But this is variable, and it may even dip abnormally to the northwest. The strike may also vary considerably with the direction of the ridges. The hanging-wall is a hard, talcose slate, the foot-wall being a hornblende schist. The ore adjoining the latter is exceedingly hard and banded to the wall. That on the hanging-wall side is softer, and, I believe, of a better grade. This hard ore was in places impregnated with crystals of pyrites.

Below I give two analyses of this ore:

	(1) Per cent.	(2) Per cent.
Silica,	14.05	11.92
Metallic iron,	53.95	57.56
Lime,	3.00	
Sulphur,	2.14	
Phosphorus,	0.068	0.008
Titanic acid,		0.520

No. 1 was obtained indirectly from Mr. C. M. Clark, of the "Virginia Steel Company," and represents an average of five samples. It also contains a slight percentage of copper and alumina.

No. 2 is by Mr. M. B. Corse, of Roanoke, Va., of a sample taken by myself.

In a northeasterly direction the outcrop may be traced within the corporate limits of Rocky Mount; and five miles northeast of Rocky Mount it is seen to cross the Lynchburg turnpike. I believe that the magnetic ores of Campbell and Amherst counties, above Lynchburg, belong to this general zone.

In the southwesterly direction the ore-zone can be traced to the Pig river, which it crosses about two and a half miles from Rocky Mount.

On the western bank of the river, at the foot of a hill on Dr. William's place, were some old pits, dug by the Sanders Furnace Company, in which the ore showed up very well. The outcrop was also observed in the turnpike a little higher up.

The old Sanders furnace and forge were situated about three-fourths of a mile from Rocky Mount. The furnace is now demolished, but in its time is said to have produced from these ores a very superior iron, famed for its excellent qualities, which was wagoned to North Carolina and Alabama. They smelted the ores mined in the immediate vicinity of Rocky Mount, and many of these old pits and diggings are still visible, though, naturally, overgrown and filled up. At one place the ore was worked in a ridge parallel to that in which are situated the Rocky Mount mines; other indications besides this lead me to believe that there are several parallel ore-veins existing in parallel ridges.

In going from the Rocky Mount to the Brown Hill section, I traced the magnetite-zone for eleven or twelve miles, paralleled by the Roanoke & Southern railroad line. Surface-indications of the ore were visible in many places. An outcrop of the vein is met with in the dirt road at the foot of Cooke's Knob, near the site of

an old furnace, the ruins of which are still standing, on the waters of Story creek. In the railroad cut, on John Young's place, near here, there were some fair indications of magnetite. The vein crosses the dirt road to the south of here, and passes over a hill on Alex. Ingram's place, where the outcrop covers a width on the surface of from fifteen to twenty yards.

Southwest from here, on Mr. Key's place, I saw quite a marked outcrop of brown hematite, in a wash-out, seemingly a superficial or blanket-formation; it showed but a few inches in thickness. The blue shales here rather indicate the possibility of a magnetite vein.

II. The Brown Hill Section.

These deposits are on the waters of Town creek, and the Roanoke & Southern railroad parallels the zone for several miles. The peculiarity of the deposits here seems to be a capping of brown hematite ore over the magnetite vein proper, which had already been noticed on Key's place. While this brown ore is presumably of very good quality, the underlying magnetite must be looked to for the main source of supply.

On Armistead Young, Jr.'s, place, where the railroad crosses the summit, are several test-pits and a little prospect-shaft, fifteen or twenty feet in depth. Unfortunately the vein has not been thoroughly exposed. The strike was between 40° and 50° northeast and southwest, and the dip measured about 70° southeast.

As I had expected, the brown hematite showed near the surface, the magnetite being below. There were some very good indications of magnetite over the side of the hill here.

To the southwest from here, on Lewis Young's place, are very profuse surface-indications of brown hematite on the slope of a ridge, just above the Roanoke & Southern R. R. line. Some large lumps and boulders of hematite were observed here, weighing probably several hundred pounds each.

In a southwest direction the zone crosses over Giles Young's place, where there are some old furnace-workings, exposing a fine ledge of magnetite. Comparatively speaking a good deal of ore has been mined here, and, nearly as I could judge, the vein must be some 6 or 8 feet thick. Strike northeast and southwest; dip, nearly vertical. The ore was a black, compact magnetite. I regret that the analysis of a sample I took here has not yet been completed.

Thence the zone crosses Mr. Spencer's land, where several small pits show the brown hematite and magnetite "bloom." Here there

are also signs of some former workings, open cuts and tunnels, now caved in; but the old dumps show that they were chiefly intent upon getting the superficial hematite-ores.

Outcroppings are again met with on Mr. Samuel Prelliman's place, in a small branch. There are some old workings here, where the Sanders' Furnace Co. mined ores. They exhibited both brown hematite and magnetite.

The following analysis represents the quality of the superficial brown ores in this section:

	Per Cent.
Silica,	15.41
Metallic iron,	48.94
Phosphorus,	0.321

Analyst, M. B. Corse, Roanoke, Va.

At Prelliman's, the railroad leaves the ore-zone, which continues in a southwest direction to Stewart's Knob, in Patrick county, 12 miles from here. It crosses Smith river, passing from Franklin into Patrick county, just above the old Hairston forge, which was in operation only 20 years ago, and the ruins of which are still standing. On the south bank of Smith river, near Union Church, the brown hematite crops out in a steep hill. An analysis of this ore, made by Mr. Corse, shows:

	Per Cent.
Silica,	5.67
Metallic iron,	53.06
Phosphorus,	0.559

From here to Stewart's Knob, distinct traces of the vein are met with, crossing the dirt road in many places.

III. The Stewart's Knob Section.

Stewart's Knob is one of the foot-hills of Bull's Mount, an outlier of the main Blue Ridge chain, in the northeast part of Patrick county.

Here are situated the old Hairston mines, at which there are a great many openings and an old tunnel which I examined. It is again a source of regret to me that at none of these places was I able to determine the real width of the vein, as the diggings and tunnel were overgrown and fallen in. However, I saw enough to satisfy me that there is a large deposit of ore here; and I believe that the veins have, in a measure, been concentrated in this knoll. The

brown hematite was again seen to overlie the magnetite. In places the ore is specular (though still magnetic) with a distinct red streak, fine-grained and homogeneous. Again it is crystalline, coarse-grained and highly magnetic, having a black streak. I am pleased to say that I did not notice the presence of any pyrites.

It will thus be seen that I traced the ore-zone from Rocky Mount in a general southwest direction, to Stewart's Knob, a distance of over 24 miles. From Rocky Mount to Prelliman's the Roanoke & Southern railroad practically parallels the zone. Continued in a southwest direction through Virginia it would strike the Cranberry deposit in Mitchell county, N. C.

The Brown Hill section is well timbered with white, post and chestnut oak, poplar and hickory.

IV. The Davis Mill Section.

On the Hollingsworth place, near Mill creek, about three miles from the confluence of the same with Town creek, there are two small test-pits some 4 or 5 feet in depth, and 10 or 15 feet long, exposing a vein of magnetic iron-ore, with the characteristic showing of limonite near the surface. The strike measured a few degrees north of east and south of west, dipping nearly vertically. In the dirt road near here there are indications of two separate parallel veins. In an easterly direction it crosses over to an adjacent ridge, where it is laid open by another pit; it should strike the railroad in the vicinity of Davis's Mill on Town creek.

About 1 mile south from here, also in the waters of Mill creek, I found an excellent outcrop of hematite ore in the dirt road, the surface showing covering a width of about 30 feet, with indications of magnetic "bloom."

About $\frac{1}{2}$ mile still further south there are similar indications of another vein. About 100 yards east of the railroad, near Davis's Mill, are some indications of brown hematite (as yet unexplored), which seem to be in the zone with the Hollingsworth place, above mentioned.

About $\frac{3}{4}$ mile from Davis's Mill on a little branch, which is a tributary of Town creek, are some very fair indications of magnetite. On the outcrop, the ore was disseminated through quartzite. Near here there were also indications of limonite and pyrolusite.

On the East prong of Town creek, 3 miles from the Roanoke & Southern railroad, there are some very good indications of magnetite. On the Martin place there were several shallow pits, not deep

enough to expose the vein, but the surface-indications were very profuse. The strike was nearly east and west, and the dip measured from 60° to 65° a little east of south.

An analysis of this ore by Mr. M. B. Corse, of Roanoke, Va., shows:

	Per Cent.
Silica,	0.99
Metallic iron,	69.87
Phosphorus,	0.010
Titanic Acid,	trace.

About 3 miles south of Waidsboro, and 3 miles east of the Roanoke & Southern railroad, on Hurt's Mount, the most prominent peak in the vicinity, there is an excellent showing of magnetic iron-ore. Some of the surface boulders of pure magnetite weighed several hundred pounds.

V. The Martinsville Section.

About 7 miles southwest from Martinsville, on the waters of Grassy creek, are the Chestnut Knob magnetites. Chestnut Knob is a prominent peak near the boundary line between Virginia and North Carolina, and the Roanoke & Southern railroad line crosses the State line about two miles east of it. Several old openings have been made in this vicinity, and at one old bank the ore was worked on a small scale as long as 100 years ago.

The surface indications were good, and the zone has been traced for some distance to the southwest. I believe that it will correspond with the ore-zone at Danbury, N. C., to be described hereafter.

The character of the ore is massive, coarse, granular, and highly magnetic.

In a northeast direction the zone can be traced by numerous outcroppings.

On the Morris place, about 4 miles from Martinsville, on the west bank of Smith river, on the Danville & New River railroad, and about ½ mile from the Roanoke & Southern railroad, there are some 5 or 6 pits on the top of a ridge, showing up the ore very well, though they are hardly deep enough to have penetrated the vein in its normal position. The ore is compact, granular and highly magnetic. The wall-rocks are a garnetiferous gneiss and a talcose mica-schist. The strike is northeast and southwest.

On the east bank of the river the zone can be traced up Little Reid creek. The surface indications at places are excellent, surpassing anything I have seen yet. Some of the boulders of magne-

tite weigh several tons. The ore is fine-grained and compact, free from quartzite and sulphuret.

Both to the northwest and southeast of this vein, the surface indications point to the existence of parallel veins.

Near Pedigo meeting-house, 3½ miles northeast of Martinsville, and about 7 miles from the R. & S. R. R., on Beaver creek, there is an exceedingly fine showing of magnetite. The ore is compact, fine-grained, hard, black magnetite; it is very homogeneous and free from quartzite and sulphuret. The surface showings are very profuse and indicate two parallel veins. There are several small test-pits here, in one of which the vein shows up about six (6) feet in width. The wall-rocks of the vein are probably quartzite and soapstone. The zone can be traced in both directions from here for some distance; and to the northeast, I think its continuation will be found in the Turkey Cock Mountains, about 3 miles distant. Magnetite and specular hematite have been reported from here.

About 1 mile north of Martinsville there is a showing of magnetite near the R. & S. R. R. An analysis of this ore shows 58 per cent. metallic iron. It would require months to see everything thoroughly in this section, but from my limited examination I feel safe in saying that there are large and valuable deposits here, which I believe to correspond with the Pittsville ore-zone in Pittsylvania county, about 34 miles northeast of Martinsville, and with the Danbury zone in Stokes county, N. C., some 25 miles to the southwest.

VI. The Danbury Section.

Danbury is situated on the south bank of the Dan river, in Stokes Co., N. C., about 11 miles northwest from the R. & S. R. R. at Walnut Cove. The town is located on the foot-hills of Sauratown Mountains.

On the north bank of the river are a series of parallel ridges, the spurs of which make in the direction of the river, being cut by numerous streams and hollows.

Here are situated the deposits of magnetic iron-ore which have been known to exist, and from which the ores have been mined and smelted, as long as 100 years ago.

The geological formation belongs to the Archæan or Eozoic, and the commonly occuring rocks are sandstone, granite, schists (mica, talc and hornblende). The stratification is very much contorted, showing evidence of some great disturbance in the earth's crust, probably

brought about by the upheaval of Sauratown Mountains and contiguous ranges.

I found the strike to vary from due east and west to due north and south, often northeast and southwest (normal) and sometimes northwest and southeast.

The dip naturally varies accordingly, as does also the angle of the same, which is usually between 20° and 40°. The magnetic ores in this section occur in a series of parallel veins, occupying a width of about 4½ miles. So far, but few of these veins have ever been prospected or worked. They have been traced in a southwest direction for over 20 miles through Stokes county, crossing the Yadkin river into Yadkin county. In a northeast direction they most likely correspond to the Henry county, Va., zone, as previously mentioned.

The ores in this district may be divided into three classes: Hard ore, soapstone ore and sand ore. This is the vernacular nomenclature, and is meant to be descriptive of the characteristics of the ore.

The hard ore is a hard, compact, crystalline, massive and homogeneous magnetite; the soapstone ore is a soft, greasy ore, composed of crystalline grains of magnetite, disseminated through a matrix of talcose and mica schists; and the sand-ore is composed of granular magnetite disseminated through a sandy matrix. It is also soft and friable.

The soapstone and sand ores taken *en masse*, are, of course, lower in metallic iron than the hard ore, which should average between 55 and 60 per cent. But they may easily be separated from their matrix by the use of the modern improved magnetic separators or concentrators, such as the Edison, Wenström, or Ball and Norton. They have the advantage of being more easily mined than the hard ores, on account of their softness.

I will now describe some of the old workings which I visited:

The William Nelson Hard Ore Bank.—Ore was formerly mined at this place for the old Clement's forge, which was built in 1790, about 1½ miles west of the ore bank. The main opening is situated in a ravine near Mr. Nelson's house, on a small branch, a tributary of Buck Island creek, which empties into the Dan river about 2 miles from here. This opening is in the shape of an undercut along the outcrop of the vein; unfortunately it was filled with water at the time of my visit. It is about 100 feet in length and shows a clearly defined vein of very hard, steel-gray to black, crystalline magnetite, homogeneous in structure and highly magnetic. The

strike of the strata was nearly east and west, and their dip measured from 30° to 35° nearly north.

The hanging-wall is a hard, gneissoid quartzite (metamorphic sandstone); the foot-wall being a hornblende schist, banded to the ore by a layer of epidote.

At the upper end of this cut I determined the vein to be 2½ to 3 feet thick, and I was told that it had been cut in the middle at a thickness of 8 feet. The outcrop can be traced in an easterly direction for half a mile or more.

On the opposite side of the branch, and on the northern slope of School-house ridge, were a number of old openings and diggings, from which hundreds of tons of ore have been apparently taken in former times. The character, strike and dip of the strata, are the same as on the opposite side, just described, and it must be the same vein which lies on the slope of the hill here in the shape of a thick blanket, and is cut in two by the branch at the bottom of the ravine. In one place a large area has been stripped and cleaned off, exposing a naked bare back of ore.

There is an old open cut here, 40 to 50 feet deep at the face, showing up the ore-vein at a considerable depth. The pit was filled up so that I could make no measurements.

The following is an analysis of a sample taken from the Hard ore-bank. The sample contained some of the wall-rock, accounting for the high percentage of silica.

	Per cent.
Silica,	17.83
Metallic iron,	53.24
Phosphorus,	0.052
Sulphur,	0.023
Titanic acid,	trace.

Lee Nelson's Grandfather Bank.—This old ore-bank is situated on Island branch, which empties into the Dan river about ¼ mile from here. The opening is in the shape of a large cut, extending into the hillside some 75 feet, and is 20 to 30 feet deep at the face. It is a reddish, talcose ore (soapstone class), soft, and is easily mined.

I was unable to make any satisfactory measurements, as the face was caved in and the drift filled up. The width of the vein is said to vary from 3 to 9 feet. The foot-wall will be a hornblende, and the hanging-wall a talcose mica-schist.

An analysis of this ore shows:

	Per cent.
Silica,	21.47
Metallic iron,	47.23
Phosphorus,	0.081
Sulphur,	0.006
Titanic acid,	0.19

The strike is nearly north and south. The dip measures about 45°, a little north of east.

Becky Nelson Bank.—This old bank is on the opposite side of the ridge, west of the "Grandfather" bank. The direction of the old openings along the surface, several hundred yards in extent, shows the strike to be northwest and southeast. There is an old tunnel here in which the vein was said to be 8 feet thick when they stopped. The ore belongs to the "soapstone" class. These old ore-banks supplied the charcoal-furnace at the mouth of Snow creek, 2½ miles below Danbury, about 60 years ago.

Southwest from the above, on the same ridge, is the *Banner Bank.* This bank shows many old pits and surface diggings, extending along the line of outcrop for several hundred yards. The vein is reported to have been 8 feet thick.

The old *Langford Bank* is about ¼ mile southeast from the *Grandfather*, and the *Shropshire Bank* is also in this vicinity.

The Rodgers Ore-Bank.—This is situated on a small branch, a tributary of Sandy creek, which empties into the Dan river about one mile from here. There is a shaft here about 60 feet deep, which was filled with water at the time of my visit. It was last worked during the Civil War, to supply the old charcoal-furnace on the river just below Danbury. The dip, measured near the surface, is 21°, slightly north of east; consequently the strike is almost north and south.

As the vein was gone down on, the angle of dip increased, as is shown by the fact that the shaft, sunk very near the outcrop on the surface, was in ore at the bottom. This folding is noticeable at other places in this district. At a depth of 60 feet levels were driven off along the strike of the vein, which is reported as varying in thickness from 4 to 8 feet. The old ore-dump outside of the shaft shows some very good specimens of hard, compact magnetite. The foot-wall is a hard, blue flint, and I infer the hanging-wall to be a hornblende schist, with a thin parting of micaceous shale adjoining the ore.

Southwest from the above place is the old *Cherry Tree Bank*, on Sandy creek, about ½ mile above the Danbury ford at the river.

The Carlin Ore-Bank.—This is situated on a ridge above a small branch, a tributary of Seven Island creek, which empties into the Dan river 1¼ miles from here. There are several old shafts here, one of which has caved in, and with it a considerable area over the old underground workings. The shaft was 60 feet deep. The ores were formerly worked in a neighboring forge.

The Pepper Mine.—This is another of these old ore-banks, at which nothing more than the former workings can now be seen, which are of considerable extent. A tunnel here is said to have cut the vein from 6 to 8 feet in thickness.

The Isaac Fagg Opening.—At this place there are several test-pits on the summit of a ridge showing magnetic sand. The main opening is at the head of a ravine, where the ore was formerly worked. At the bottom of this opening a drift was extended into the face some 6 or 8 feet, exposing a vein of about 4 feet thickness. It is a sand ore, soft, and easily mined. The dip is 21° southeast.

An analysis of this ore shows:

	Per cent.
Silica,	18.35
Metallic iron,	48.62
Phosphorus,	0.117
Sulphur,	0.021
Titanic acid,	0.17

In several instances I noticed indications of iron-ore on the south bank of the Dan river. In the Sauratown mountain range, surface-indications of both magnetite and brown hematite have been observed, though they have never been prospected as to exact location and quantity.

Discussion.

Edmund C. Pechin, Roanoke, Va. (communication to the Secretary): Mr. Nitze's paper is interesting, and in one sense of the word, valuable, inasmuch as it calls attention to a comparatively new district, about which we know little or nothing. Its drawback is that it is cursory and of little practical value as giving definite information; but this the author frankly admits.

It is to be regretted that he could not have had the opportunity of more fully and exhaustively examining the section in question, so as to make a full and satisfactory report concerning it, like that which he has made on the Pittsville workings in Franklin county, Virginia.

My purpose in commenting on this paper at all, is to call attention

to the grave possibility of serious defects in the quality of these magnetic ores, and the still graver possibility of deficient quantity. I must be permitted to call attention to one important fact which is constantly coming under my notice, the neglect of the prospector to secure *average samples* for analysis. There are thousands of published analyses of Southern ores which are wholly misleading, because they represent, not the average composition of an extensive deposit, but that of exceptional and, very frequently, meager portions. I am forced to this conclusion as regards Mr. Nitze's samples, from the fact that he gives returns from ores taken from a working which has been under my own careful personal supervision, and subject to very cautious consideration and sampling, and which gave results widely differing from his. On the property described by him in the Rocky Mount Section as having been worked in the past by the Pennsylvania Steel Company, there are two distinct parallel deposits or zones of magnetic ore, about three hundred feet apart; one running with the crest, and the other at the foot of the hill. The shafts on the upper one are all above water-level; on the latter, all below water-level.

Last summer several of these old shafts were unwatered and sunk deeper, two on the upper and one on the lower vein. Both veins pinched and bulged, and at the surface were from 5 to 7 feet wide. The upper one, dipping at the surface from 60° to 80°, suddenly flattened at the depth of 45 feet to a very slight dip. The iron-ore was badly mixed throughout with hornblende, and was so unsatisfactory on a fair sampling and so hard and expensive to mine, that these workings were abandoned. I very carefully sampled the stock-piles and the bottoms of the shafts, to ascertain the value of the ore which would have to be mined and shipped, in regular working, and the following was the result:

Upper Vein.

	No. 1 Shaft. Per cent.	No. 2 Shaft. Per cent.
Metallic Iron,	44.40	39.06
Silica,	20.94	25.44
Manganese,	.04	.06
Phosphorus,	.213	.245
Sulphur,	3.20	4.06

Attention was then turned to a shaft on the lower vein, which was unwatered and deepened. In this vein the ore was better as to iron and silica, and had a thickness of about 5 to 6 feet, separated by 2½ feet of hornblende. This hornblende proved to be a wedge, gradu-

ally thinning in depth, until the two bands of ore came together, but at this junction the ore began to thin also, and at 75 feet entirely pinched out, the foot and hanging-walls coming together. No decided attempt was made to sink deeper, as the character of the ore did not warrant a further expenditure of money. A particularly careful sampling was made at this shaft. Unquestionably the ore nearer the foot-wall was richer and freer from rock, but the high percentage of sulphur discouraged any more prospecting. In the analyses from this and other properties made by Mr. Corse, and published in Mr. Nitze's paper, it is evident that the chemist was not instructed to determine sulphur, which, however, the analysis credited on page 176 to the Virginia Steel Company as an average of five samples, shows to be present.

The result of our sample-analysis on the lower vein was as follows:

No. 1 Shaft, Lower Vein.

	Next to Hanging-wall.	Next to Foot-wall.	Stock-Pile.
	Per cent.	Per cent.	Per cent.
Metallic iron,	49.08	51.60	49.50
Silica,	15.79	14.30	16.29
Manganese,	.08	.08	.08
Phosphorus,	.200	.172	.184
Sulphur,	4.85	3.81	4.83

As a matter of curiosity, I took a sample of the hornblende rock dividing the ore in this shaft. To one unfamiliar with mining, when broken down it looked like the ore. It was found to contain, on analysis by Mr. Walter Graham, the following:

	Per cent.
Metallic iron,	19.92
Silica,	39.04
Manganese,	.06
Phosphorus,	.067
Sulphur,	.90

If this hornblende had kindly insisted upon appropriating the phosphorus and sulphur distributed through the ore, we should have been glad to have it continue indefinitely (that is, if the ore had continued also), even if it had added somewhat to the cost of mining.

After this positive experience one is compelled to look with distrust, not only upon this particular zone as it continues southwestwardly, but upon the other exposures reported.

The pits and openings scattered throughout the country, which

served as the sources of supply for the old charcoal-furnaces, amount to nothing as evidence. Nearly twenty years ago a charcoal-iron-maker from Patrick county, Va., gave me a sample of a superb, high-grade magnetic ore, which he said he was using, but there was nothing to indicate that he had at any one point upon his large estates a large workable deposit of it, except the fact that he secured his limited supplies from surface "float" and shallow openings.

There is only one way to test these deposits, viz., by the diamond-drill. If the veins, even though narrow, can be found in depth, and the ores are of the quality as represented, they will prove of large value to the owners and of inestimable benefit to the whole district, either for working by themselves or as admixtures. But we want *average*, not exceptional samples.

SINKING THROUGH WET GRAVEL AND QUICKSAND NEAR NORWAY, MICH.

BY WILLIAM KELLY, VULCAN, MICH.

(Cleveland Meeting, June, 1891.)

AN extensive swamp covers a large part of the town-site of Norway and adjacent land. Through this swamp run two parallel ore-formations. In the north one is the Aragon mine. The south one passes into the swamp on the Harrison property. On the edge of the swamp, about 1000 feet from the Aragon mine, a diamond drill, in the fall of 1889, located the ore-bearing formation and a probable ore-body. To explore further and develop this ore, the Penn Iron Mining Company proposed, in the spring of 1890, to sink a shaft.

The drill had shown the depth of the glacial drift to be more than 60 feet. A test-pit near the location of the proposed shaft had struck, at a depth of 20 feet, an amount of water which a 200-gallon pump was unable to lower. At the Aragon, a few years before, a shaft had been sunk to hard pan with great difficulty and many delays by the usual method of driving laths.

Under the conditions it was decided to sink a caisson- or drop-shaft. Two 40-horse-power boilers, a Lidgerwood engine with 4-foot drum, and a good derrick were set up; and two No. 10 Knowles

pumps, rated at 400 gallons a minute, were brought on the ground. The pumps had been purchased for another purpose, and, being brass-lined, were not well suited for the work intended, but they were available.

The dimensions adopted for the top of the shaft were 6 feet by 13 feet inside. To give sufficient space for pumps and working, and to aid the shaft to settle, it was made 4 feet larger each way at the bottom. The shaft was divided, to within 12 feet of the bottom, into three compartments, the middle one uniformly 4 feet wide. This compartment was used for hoisting, a ladder-way and pipes. The pumps were placed one in each end-compartment. Above the pumps the end-compartments were planked up to be filled with sand to increase the weight. A ventilation-box was put in one corner. The bottom of the shaft was left unobstructed for working-purposes, and sufficiently high to allow two additional pumps to be put in under the first. (See Fig. 1.)

The bottom-pieces, made of oak and constituting what is called the shoe (see A, A, Fig. 4), were 15 inches square, but the bottom inside was beveled off to 6 inches. Above the shoe, white-pine timbers (B, B), 12 inches square, framed in sets, were laid close and bolted together and to the shoe with eight 5-foot bolts. The successive sets were reduced 1 inch in length and width, until at 48 feet above the bottom their dimensions corresponded with the top. Corner-posts, 12 inches square, of unequal lengths so as to break joints, were bolted to every other side-piece and end-piece. The bolts, being put in from the inside and having the nuts countersunk, were easily unscrewed and recovered when the corner-posts were removed. Like the corner-posts, side-posts (D, D) were put in, one at each corner of the middle compartment. Twelve-inch dividers (E) were used every 5 feet.

After the ground had been leveled the timbers were built up and bolted as far as the derrick and bucket would permit, nearly 30 feet. The seams were then carefully caulked outside, and 3-inch planks (C) in unequal lengths were spiked on, to protect the caulking and timbers and to strengthen the shaft. A shaft of this character is liable to be pulled apart if not very strong.

Steam-hose was used at first to connect the pipe on the ground, coming from the boilers, with the pipes in the shaft. But afterwards lengths of pipe with double elbows were substituted. Two lines of steam-pipe from the ground to the pumps were necessary, in

order that one could be used while the other was being altered in building the shaft up.

The shaft erected, the pumps in place and everything ready, ground inside the shaft was broken Monday morning, June 2, 1890. As the top was dry, the buckets came up fast, and by next morning the shaft had gone down 6 feet. On Friday, the fifth day, at 15 feet, the pumps had to be started. The first week's work resulted in 18 feet sunk. During the first three days of the second week 9 feet more were sunk. It was now evident that both our pumps had to run fast to keep the water out, and if one should break down or the water should increase, we would be drowned out. Therefore, before sinking the pumps below the water-level, we stopped to get more power.

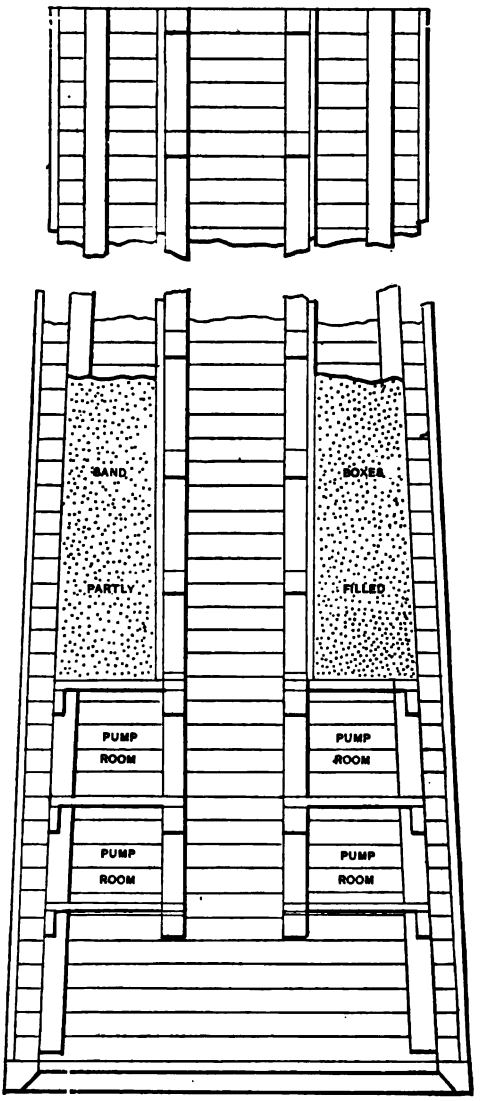
Two portable boilers, of 35 and 100 horse-power respectively, were bought, delivered and connected, and two No. 10 Cameron pumps without air-chambers were placed 4 feet under the Knowles pumps. During the stop the shaft was built up again as high as possible. We lost thus 15 working days, but started afresh Monday, June 30th, with doubled power. During the next three days we sunk 7 feet. Then the Knowles pumps gave out. Eight days of sand and gravel had completely worn out the brass linings. It was necessary to take out the linings and put in larger pistons, which repairs consumed 2 days.

On Saturday, July 5th, we started again, and to the end of the following week we sank 16 feet in 7 working-days, during which time everything went well. The pumps were kept busy; three running constantly, and the Knowles pumps often making 160 strokes a minute. The quantity of water was estimated at 1500 gallons a minute. During the week the sand-boxes were filled to keep the shaft down to the bottom of the excavation. The sand and gravel came in under the shoe, and the surface about the shaft settled into a large pit which continually grew larger.

July 12th, the shaft was down 50 feet, and it became necessary again to build it higher. This took 3 days. A drill in the bottom gave us some encouragement, as at 10 feet it struck something hard. During the next 3 days we sunk 7 feet and found hard pan in a corner of the shaft. At this point the shaft did not settle regularly. Sometimes it would hang, and then it would drop with a great inrush of sand and water, even when the ground was out a foot or more from under the shoe. To increase still further the weight of the shaft, 30 tons of rails were laid on the top; and props had to be

placed against the shaft at different times to keep it straight. It took 18 days to go through the 14 feet of hard pan; but parts of

Fig. 1.



LONGITUDINAL SECTION
Scale ½ in. to 1 foot
HARRISON SHAFT

two days were spent in weighting the shaft and one day with an accident which bade fair to stop proceedings summarily.

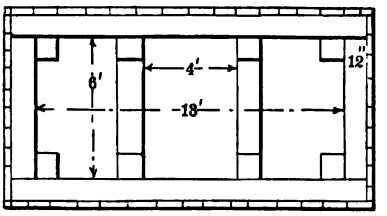
Some difficulty with the water-supply of the boilers cut down the steam-pressure so low as to stop the pumps, and before steam was raised again they were completely drowned and could not be reached. An anxious quarter of an hour was passed before first one, then another and then a third pump started of their own accord under water.

The ledge was struck August 7th, and two days later the slates were exposed all across the shaft. Four days more were spent getting the shaft down 2 feet further into solid rock.

The time spent in sinking may be summarized as follows:

4 days sinking 15 feet above water-level;
17 days sinking 42 feet through wet gravel and quicksand;
16 days sinking 14 feet through hard pan;
4 days sinking 2 feet in slates; making a total of
41 days sinking 73 feet. To this must be added

Fig. 2.



PLAN OF TOP SET
Scale ¼ in. to 1 foot
HARRISON SHAFT

6 days required to build up the shaft and
2 days weighting shaft with rails, making a total of
49 days, or one day over eight weeks actual working time. In addition to this we lost
12 days in increasing our power and
2 in repairing pumps, making the time consumed
63 days, or ten and a half weeks.

We now, August 14th, had the shaft down firmly in the ledge; but the most delicate part of the operation was still to come, namely, stopping the flow of water. Before that could be done, however, many things were necessary.

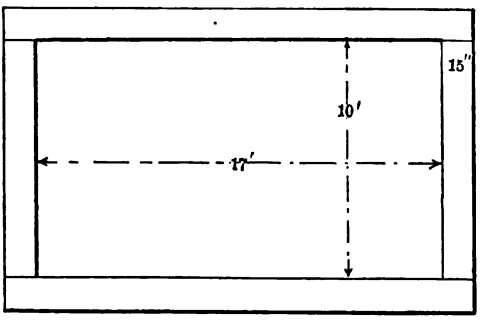
The rails had to be removed from the top and the sand from the boxes, the pipes changed and the shaft built up to the surface. There was now a sink-hole about the shaft, 65 feet in diameter and 20 feet

deep, and the top of the shaft was about 6 feet below the original surface-level. The shaft was but little out of plumb, the top set having to be raised 2 inches at one end to level it. The corner-posts were taken out, the bolt-holes plugged and the shaft caulked on the inside. This work took 8 days.

The next 14 days were spent in sinking 11 feet further into the ledge. It was unsafe to make heavy blasts, and the pumps, after their severe strain with lengthened suctions, were barely equal to their task. The bulk of the water from under the shoe was gathered together by troughs of boards and clay, but still water was falling everywhere.

The work of sealing up the bottom of the drop-shaft was now undertaken. A set, 6 feet by 13 feet inside, of 12-inch square timber, was carefully placed in line with the top set of the shaft, about 6½

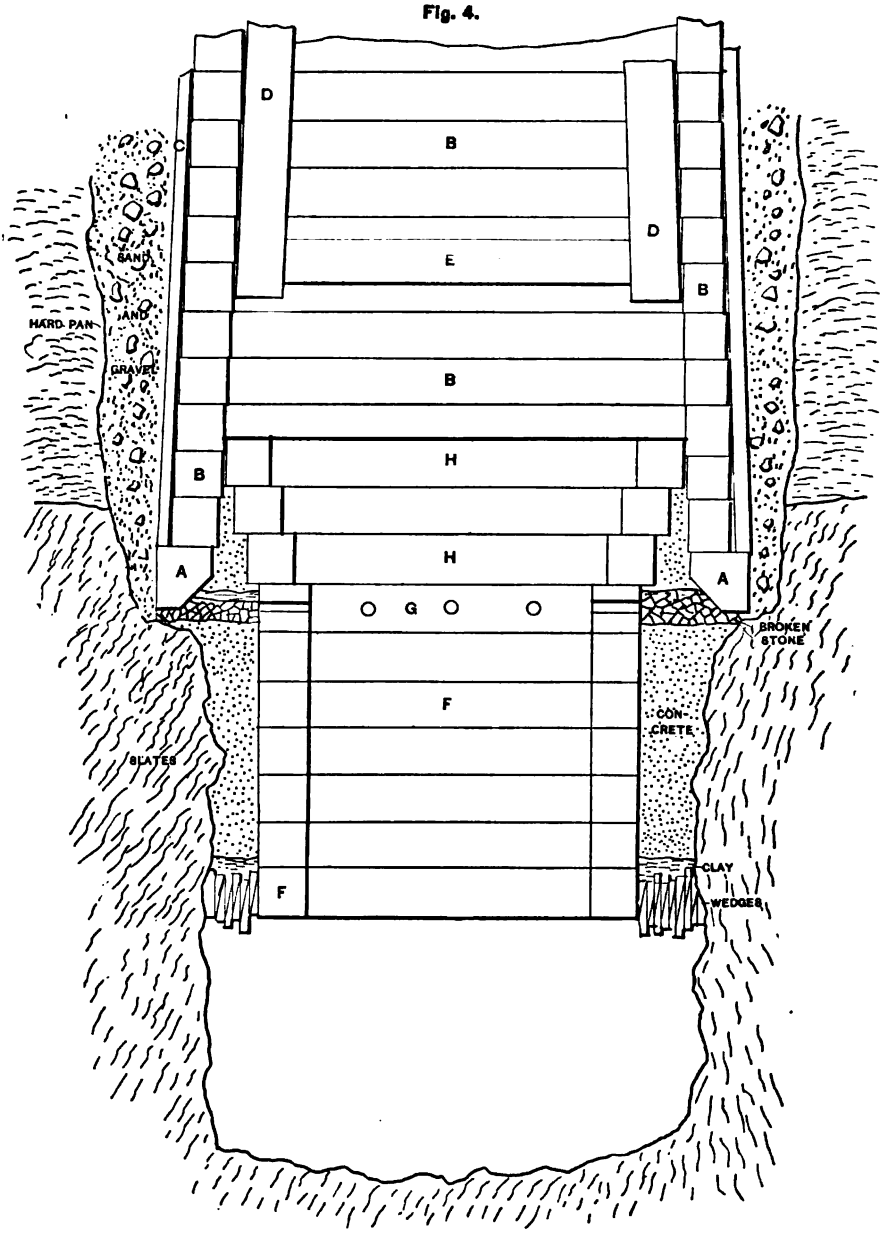
Fig. 3.



PLAN OF SHOE
Scale ½ in. to 1 foot
HARRISON SHAFT

feet below the shoe. This was thoroughly blocked against the rock all around with wedges. Six sets (F, F) of the same size were placed on top of the first and each bolted to the next. Behind the sets as they were built up, was put a thin layer of clay over the wedges and then concrete of equal parts of sand and Hilton cement. The middle of the top set (G) was about opposite the bottom of the shoe. Through this set, twenty 2-inch holes had been bored. Behind the holes a layer of gravel and broken stone, 4 inches deep, was laid, leaving a free passage for the water. Upon this perforated set were put three other sets (H, H) of increasing inside dimensions, so that the top set was against and bolted to the drop-shaft. The space behind these sets was filled with concrete as before. This timbering and cementing in such a flood of water was a tedious process and took 18 days.

Fig. 4.



CROSS SECTION

SHOWING METHOD OF CLOSING THE BOTTOM

Scale ¼ in. to 1 foot

HARRISON SHAFT

It was all done, and we were ready to plug up the holes, when the pumps failed. One had been removed to make room for lowering the timbers. The piston-rod of another broke and a third would not draw. In a few minutes the pumps were flooded and the water gradually rose to its old level. Then followed ten days in recovering the shaft by means of other pumps. The perforated set could not be cleared of water, as the broken pump was the lowest and the others could not be got down low enough. The first few holes were plugged without difficulty under two feet of water, but the last one defied us for some hours until we used a long plug reaching across the shaft.

The result was entirely satisfactory. The water at once fell to about 200 gallons a minute. After the pumps and side-posts had been removed and the interior had been thoroughly caulked, the water was decreased to about 90 gallons. After the shaft had been sunk further and bearers put in, a small station was cut at one end and the water was gathered to a No. 8 Cameron pump. Below this the shaft was sunk with a No. 4 Cameron which now works about an hour and a half a day.

The time taken for sealing up the bottom may be summarized as follows:

8 days to alter shaft after it rested;
14 days to sink 11 feet in slates;
18 days to timber and cement;
16 days to remove pumps, caulk and arrange shaft for regular sinking, making a total of
56 days. To this we must add
10 days lost by failure of pumps, making a total of
66 days or eleven weeks.

This makes the total of both periods of the work 129 days or 5 months. At the end of this time, November 1st, the shaft was 84 feet deep and in shape for sinking in the regular way.

THE FIRST IRON BLAST-FURNACES IN AMERICA.

BY W. H. ADAMS, M. E., NEW YORK CITY.

(Cleveland Meeting, June, 1891.)

SHORTLY after becoming one of the van-guard of mine-developers in the State of Virginia, during the year 1883, I called the attention of the Institute to certain deposits of pyrites, which have been largely worked since.

In all cases these deposits are covered with iron-ore (limonite), extending 40, 50 or 60 feet from surface to water-level.

The several classes of brown iron-ores are found almost continuously along a prominent ridge of mineral-bearing slates, which can be traced from the James river northeasterly to the North Anna, the Rappahannock and the Potomac rivers, crossing the counties of Louisa, Spottsylvania and Stafford. Outcrops are plainly discoverable from end to end of this belt, also, in places, to the right and left of it, and at intervals there are evidences of workings, some of them for known purposes and others of unknown age and meaning.

These old workings have an interest aside from the minerals they may contain or may have yielded. I have been engaged during the past five years in the collection of facts concerning them, which might determine definitely the site of those iron-furnaces, placed by tradition within reach of "the towne called Germanna," which were the first ever built in this State or on this continent.

The events which led to the establishment of this important industry are sufficiently narrated in a paper written by Mr. R. A. Brock, Secretary of the Virginia Historical Society, and printed in the *Proceedings of the National Museum*, 1885, page 77, from which I quote the following:

"To Virginia, the first of the English settlements in America, belongs the honor of inaugurating within her limits, as a colony, that most important industry, iron manufacture.

"The London Company contemplated a variety of manufacturing enterprises from the very beginning of its authority; prominent among them was that of iron. In 1610 Sir Thomas Gates testified before the council of the Company at London, that in Virginia 'there were divers minerals, especially iron-oare,' lying upon the

surface of the ground, some of which, having been sent home, had been found to yield as good iron as any in Europe.

"Under a new administration of its affairs, the London Company, in 1619, after twelve years of unprofitable expenditure, sent to Virginia a large body of emigrants, including workmen, and materials for some new branches of industry. These embraced no less than 100 persons skilled in the manufacture of iron, with the design of erecting in the colony three iron-works.

"Of these, 110 were from Warwickshire and Stafford, and 40 from Sussex, and were selected for their skill and industry. A part of the funds liberally contributed in England about this period, for a College at Henrico, for the education of native and colonial youth, was appropriated by the Treasurer, Sir Edwin Sandys, to the erection of iron-works, in the expectation of deriving a revenue from that source. Works for smelting the ore were erected in 1619 on Falling Creek, a tributary of James river, in Chesterfield county, about 7 miles below the city of Manchester. Most sanguine hopes of profits were cherished.

"Three of the master-workmen having died, a reinforcement of 20 experienced hands was sent over in 1621, accompanied by Mr. John Berkeley and his son, Maurice, as skillful persons to superintend the operations. A mine of the brown iron-ore in the neighborhood was opened and found to yield a 'reasonably good iron.'

"But the jealousy and enmity of the native inhabitants had unfortunately been aroused. In an hour of fancied security, when all suspicions of hostility had been lulled by the friendly protestations of the Indians, on the morning of Friday, March 22, 1622, a general attack was made by the savages upon the settlements of the colony, and 347 persons slain. Of those engaged at the iron-works at Falling Creek, all perished save a boy and girl, who fled to the bushes for safety.

"The iron-works being demolished, so great was the discouragement that a long period elapsed before the useful manufacture was again attempted in Virginia. A writer from the colony, in 1649, published that an 'iron-work erected would be as good as a silver mine.' The exportation of iron from the colony was forbidden by an Act of the Assembly in 1662, on penalty of 10 pounds of tobacco for every pound of iron exported; and the prohibition was renewed in 1682.

"Colonel William Byrd, the first of the name and family in Virginia, obtained, April 20, 1687, a grant of 1800 acres of land in Henrico county, on the south side of James river, within the limits of which was included the site of the fated iron-works on Falling Creek.

"On the 29th of October, 1696, he obtained a patent for 5644 acres lying contiguous thereto, giving as a reason for such action that 'there having been iron-works on Falling Creek in the time of the Company, and Colonel Byrd having an intention to carry them on, and foreseeing that an abundance of wood might be necessary for so great a work, he took up a large tract,' etc.

"He died on the 4th day of December, 1704, and it is not known that either he or his son and heir, of the same name and title, ever instituted any further steps toward the revival of the works at Falling Creek."

It will be understood that the iron-works noted above employed the modified "Catalan forge," types of which are yet to be found in this country.*

* Dr. T. Sterry Hunt objects to the term "Catalan forge," insisting that we have in this country only types or modifications of the old "bloomery process," brought from Germany.

This much of the earlier history of iron-making in Virginia makes us wish that sufficient records had been preserved to enable us to construct a consecutive narrative of all the operations of the colonists. It is certainly reasonable to suppose that the explorations, made by the several parties brought out from England at different times, must have extended over a wide area, and must also have resulted in discoveries of value.

In June, 1870, after excessive rains, the long dam on the Jones branch of the Chickahominy river, in New Kent county, was broken, with much damage to the mill and surroundings. The force of the current washed out and disclosed the remains of an early forge or bloomery, including some of the foundation timbers, bricks from a chimney, an anvil, a hammer and six bars of cast-iron, all of which can now be seen near by. The cast-iron bars and hammer bear no date, but have the letters "Deep Creek" cast on their surface. No record of such a location or furnace can be found, although "Deep Creek" might have been the home brand of some furnace in the old world. The location where the wash-out disclosed these relics is now known as Providence Forge. Tradition credits the erection of this forge to Colonel William Byrd; but no date is given, although discoveries of more than passing value must have been made to induce him to renew the realty-ownership of the Falling creek property in the year 1696, as noted above.

After a long period of silence as to the actual doings of the colonists who were brought over for the special purpose of iron-making, history takes up the thread again in the year 1714, at which time there was an importation of Germans, whose destination and *status* has been the subject of a great deal of discussion, owing to the fact that, as they were recruited by Baron de Graffenreidt, and have been spoken of as Palatines, some persons have supposed them to belong to the first colony of Palatines, brought over by him, which founded, under a patent of 1709, the town in North Carolina called New Berne, after Berne in Switzerland. A considerable number of these colonists were massacred by the Tuscaroras in 1711. But it can be proved beyond question that another German colony landed at Rappahannock in 1714, coming direct from Oldenburg, Germany; and, gathering all the data lately brought to light with regard to the doings of these people, we find that they were at once taken to the territory within thirty miles of the present town of Fredericksburg, where extensive explorations must have been made to justify the large expenditure thus incurred. What it meant to locate a colony

so far from any base of supplies will be appreciated by those who have perused the earlier history of the struggles and privations of the colonists on the James river.

We know that somewhere in the region between the falls of the Rappahannock river and the site of Germanna, there had been established, as early as 1676, a fort or outpost for protection against Indian incursions; and it is reasonable to suppose that some settlements were made between that fort and the river at Fredericksburg. Spotswood became Governor in 1710, and shortly after accomplished the memorable passage of the Blue Ridge, previously deemed impassable for the whites. No doubt, he learned during this trip, as his route was by way of the Rapidan river, the character of the country about Germanna, its ores, soils, climate, roads, etc., and it is also supposable, that among those who accompanied him on his march to the "vast mountains," there were iron-masters from the James river settlements. The experience of these iron-masters with the shallow deposits along the lower river, and the poorer classes of siliceous ores about the upper river, at Lynchburg, etc., must have inclined them favorably at once towards the section through which they travelled, as they crossed belt after belt of country showing signs of iron everywhere, and noted along the Rapidan the excellent deposits of drift iron-ore. These latter exist to-day, the deposit at the south crossing of Germanna being of notably good quality.

What more natural than the drifting of a mining community towards the territory, which, upon superficial examination, or, perhaps, after working tests by small parties, appeared to be supplied with ores so clean and tractable that old forge-processes were not thought of, and direct smelting for pig-iron was determined upon from the start?

Governor Spotswood enters into the historical record at this point in a communication, under date of 1714, to "Ye Lords Commissioners of Trade:"*

"An Act for exempting German Protestants from the payment of levies, is made in favor of several families of that nation, who, upon the encouragement of the Baron de Graffenreidt, came over hither in hopes of finding out mines. They were engaged principally in mining in their native land, but the Baron's misfortunes obliged him to leave the country before their arrival. They have been settled upon the frontiers of the Rappahannock, subsisting chiefly at my charge and on the contributions of some gentlemen that have a prospect of being reimbursed by their labors."

* *Spotswood Letters*, vol. i., page 20.

There is a conflict of authority as to the numerical strength of this colony, but the Rev. Philip Slaughter, D.D., one of the most patient investigators of Virginia history, cites* a memorial from the archives of the "English Society for the Propagation of the Gospel in Foreign Parts," which places the number of emigrants at thirty-two families, and gives the name of the minister who served them as Henry Haeger, a man about seventy-five years old.

Hugh Jones, in his *Present State of Virginia* published in London in 1724, says:

"Beyond Colonel Spotswood's furnace, above the falls of the Rappahannock river, and within view of the vast mountains, he has founded a town called Germanna, from some Germans sent over by Queen Anne. . . . . Beyond this is seated the colony of Germans or Palatines, with allowance of a good quantity of rich land who thrive very well and live happily and entertain generously."

There is a tradition that some of the Germans were massacred by the Indians during the first year; but this is evidently an error, growing out of the earlier troubles. There is no proof that the work was halted from any cause, but on the other hand, there are evidences of the prosperity of the colony in the records of little difficulties engendered over land sales, such as incidentally exclude the hypothesis of more serious embarrassments. In 1716, Spotswood was charged with transferring land in that section to his personal favorites, instead of to the Germans, as directed by the Crown. In answer to these accusations, he said: "The Germans were not insensible to the kindness he had shown them, and that, instead of their being his tenants, they might have been his servants, in view of the passage-money, 150 francs, they owed him." The books of the Register of the Land Office show that in 1722 Spotswood granted to Richard Hickman 28,000 acres of land, for a monetary consideration alone. In 1732 the same land was "confirmed" to Spotswood, the consideration expressed being a nominal sum of money and the importation into the country of certain Germans, whose names were given.

Colonel Byrd (second of the name and title), in his *Progress to the Mines*, thus speaks of Germanna:

"This famous town consists of Colonel Spotswood's enchanted castle on one side of the street, and a baker's dozen of ruinous tenements on the other, where so many German families had dwelt some years ago, but are now removed ten miles higher . . . . to land of their own. . . . . I let him understand that besides the pleasure

* *History of St. George's Parish*, page 98.

of paying him a visit, I came to be instructed by so great a master in the mystery of making of iron, wherein he had led the way, and was the Tubal Cain of Virginia. He corrected me a little there by assuring me he was not only the first in this country, but the first in North America who had erected a regular furnace. That they ran altogether upon bloomerys in New England and Pennsylvania, till his example had made them attempt greater works. But in this last colony, they have so few ships to carry their iron to Great Britain, that they must be content to make it only for their own use, and must be obliged to manufacture it when they have done. That he hoped he had done the country very great service by setting so good an example. That the four furnaces now at work in Virginia circulated a great sum of money for provisions and all other necessaries .... that they are besides a considerable advantage to Great Britain, because it lessens the quantity of bar iron imported from Spain, Holland, Sweden, Denmark and Muscovy, which used to be no less than 20,000 tuns yearly, though at the same time no sow iron is imported thither from any country, but only from the plantations. Then I inquired after his own mines, and hoped, as he was the first engaged in this great undertaking, that he had brought them to the most perfection. He told me he had iron in several parts of his great tract of land, but that the mine he was at work upon was thirteen miles below Germanna. That his oar, which was very rich, he raised a mile from his furnace, and was obliged to cart the iron when it was made, fifteen miles to Massaponux, a plantation he had upon Rappahannock river. .... He had been building an air furnace at Massaponux, which he had now brought to perfection, and should be thereby able to furnish the whole country with all sorts of cast iron, as cheap and as good as ever came from England. Col. Spotswood said the furnace might be built for £700, if I went the nearest way to do it, especially since coming after so many, I might correct their errors and avoid their miscarriages. That freight, custom, and other charges in England, comes to 27/ a tun. The whole expenses here, and in England, may be computed modestly at £3 a tun, and the rest that the iron sells for will be clear gain, which 'tis to be hoped will be £3 more. We drove over a fine road to the mines, 13 measured miles from Germanna, each mile marked distinctly on the trees,—all the land hereabouts seems paved with iron oar, so that there seems enough to feed a furnace for many ages. From hence we proceeded to the furnace, which is built of rough stone, having been the first of that kind erected in the country. Here the wheel that carried the bellows was no more than 20 feet diameter; but was an overshot wheel that went with little water. This was necessary here, because water is something scarce, notwithstanding 'tis supplied by two streams, one of which is conveyed 1900 feet through wooden pipes, and the other 60. The name of the founder employed at present is one Godfrey, of the Kingdom of Ireland, whose wages are 3/6 per tun for all the iron he runs, and his provisions. This man told me that the best wood for coaling is red-oak. He complained that the Colonel starves his works out of whimsicalness and frugality, endeavoring to do everything with his own people, and at the same time taking them off upon every vagary that comes into his head."

This evidence proves that between 1714 and 1739, the year of Governor Spotswood's death, active work was going on in what was then known as Spottsylvania county. *

No records have been found, as yet, giving the details of the work

* Orange county was divided from Spottsylvania county in 1734.

of this colony of miners and furnace workers, the extension of operations from time to time, the amount of ore mined, or of pig-iron made and sold, etc.—details which would have thrown much welcome light upon the establishment of the iron-industry on this continent.

I have endeavored, as a labor of love, to make good this deficiency of documents as far as possible, by tracing the old mines along the outcrops from end to end, thus adding, step by step, to the meager records such facts as make clear the reasons for operations in this particular direction, and also account, in a measure, for the absence of other data on this subject of really national importance.

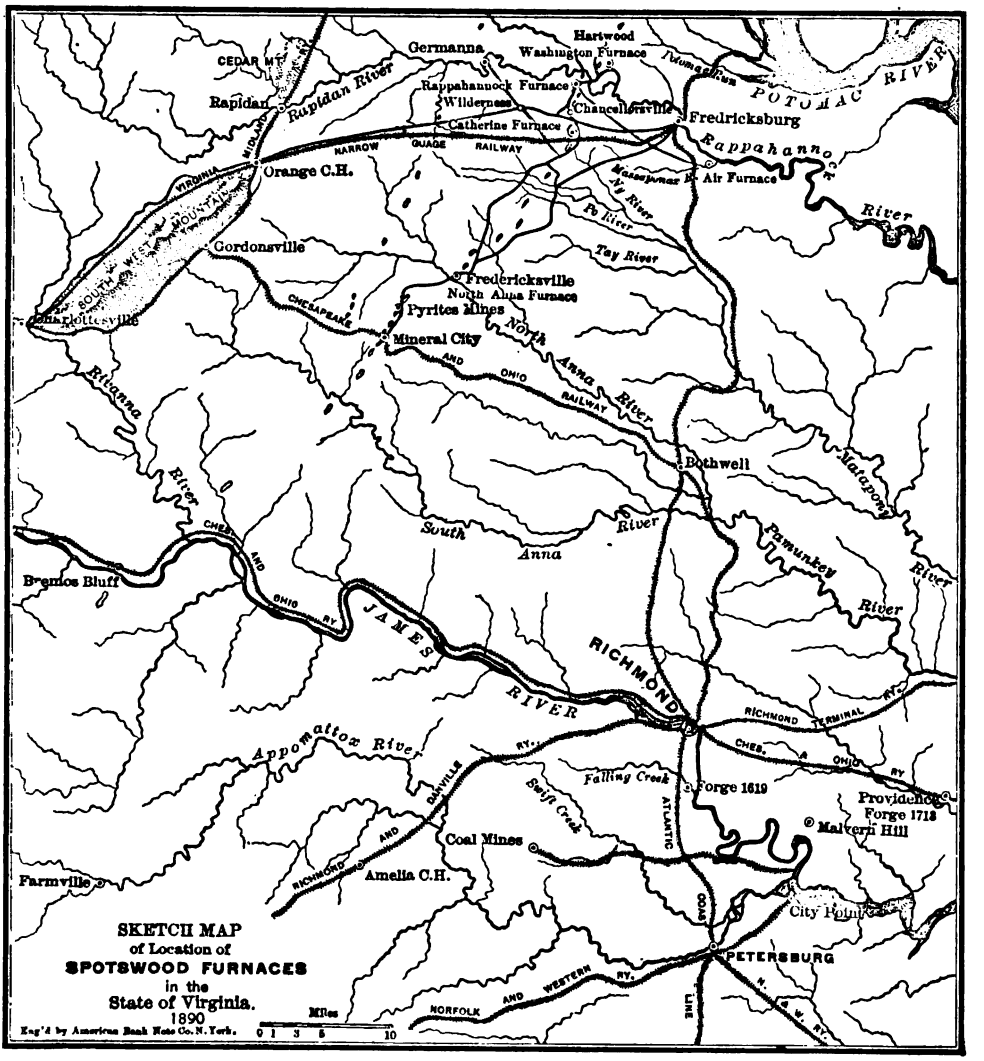
The sketch-map of the country covered by these investigations shows that the deposits of iron-ores opened by the German Palatines and their successors extend from a point north of the Rappahannock river, southwest to the present crossing of the Chesapeake and Ohio Railway, at Mineral City. The wagon-roads traced on the early maps have been of the greatest assistance to me. Many of them, though now obliterated, overgrown and fenced off, have been found, nevertheless, by careful search. Following these roads, and within easy reach of them, the openings, or iron-pits, occur, mainly along the belt of the brown ores—the variety most extensively mined, the specular ore of the western belt being more difficult to mine and transport. The openings are never deeper than 30 feet, but they range from a few feet to 1200 feet in length. Specific description of the localities will be reserved for a paper which I hope to present after further examinations shall have been completed.

Following this mineral belt by means of the roads laid out by Governor Spotswood, it is proper that we start from the town made memorable by him as the center of radiating industries and flourishing communities, for the early prosperity of which he deserves more credit than the present generation remembers to be his due.

Germanna is now but a name, and scarcely even a name, if modern maps be taken as sole authority. Originally it was the center of a royal grant of 194,000 (now reduced to less than 1000) acres. Its well-tilled fields lie snugly within the sheltering embrace of the highlands on three sides, and almost surrounded by the sparkling waters of the Rapidan river.

The fort which stood for so many years the frontier bulwark of civilization in Northern Virginia, protecting with its guns the rope-ferry or the later bridge and landing, and connected by an under-

ground tunnel with the governor's mansion, has disappeared. Its very site must be guessed at in the ploughed fields which have taken its place. The "enchanted castle" itself, a marvel of comfort, and



NOTE.—The old pits along the outcrops of iron-ore, from the Rappahannock to the James, are indicated on this map by small irregular ovals.

elegance to the travellers of those days (whose enthusiastic accounts of it emphasize also its presiding mistress, the charming wife of the governor), was long ago wasted by fire and decay. Its mate-

rials, even, have been scattered. The court-house, so long the seat of government of this vast domain, the dwellings, storehouses, mills, and shops, are all things of a past time.

One object, famous in story and song, can be seen to-day, as it was seen of old by pilgrims from far and near—the "Fountain Spring," flowing clear, cold, and abundant from a cleft in the greenstone, through a lining and into a basin composed of the clearest quartz, the purity of which gave rise to the statement that "in his extravagance, the governor even imported marble for his fountain spring." There are also some interesting relics, such as a sideboard, table, clock, portraits, etc., preserved from the fire which destroyed the governor's household, and now the property of his direct descendant, Alexander Spotswood, Esq., who lives within a few miles of Germanna.

On lands now belonging to Benjamin Sisson, Esq., several miles higher up stream, there are a few evidences of the mill and dam erected by the Palatines. Moreover, some incomplete records still exist of the business done at the town in the sale of "fire-backs," etc., and there are several of the cast-iron articles which came from the foundries, or air-furnaces, in the region; but nearly all of the best work done by these Germans must have been lost or destroyed in the changes of the century since the Revolution, or is at present in the hands of collectors.

The immediate neighborhood was the rendezvous for Indian tribes, as, in addition to the minerals and earths obtainable here for their utensils, weapons, and paints, there was the best of fishing at all seasons, and the locality was specially favorable for the curing of fish and game. This may have had something to do with the location here of a trading-post in 1670, and a fortified station for military operations in 1676, and the actual settlement of the town-site by Governor Spotswood, in 1712-14, in a position midway between the two well-known agricultural and mineral sections of this newly-opened territory.

It was not difficult to fix the localities of the furnaces which Col. Byrd describes, with more or less detail, as already in operation. These points, as shown on the sketch-map, I have verified at different times, by following the routes given by Col. Byrd, and examining the outcrops of the mineral belt. That the furnace-sites were evidently chosen with deliberation and with knowledge of all requirements, must be conceded after an examination of all the circumstances; and it is certainly remarkable that historical landmarks

established with so much intelligence should have been so easily forgotten as to make their past existence, even, matter of doubt or denial.

From all the data and from close study of the materials in place, I believe the furnaces were erected in the following order: the Rappahannock furnace; the Fredericksville furnace; the Washington-England furnace. In none of the records are more than three blast-furnaces mentioned; and there is no location on the mineral-belt which could have supported other furnaces, during the working period here under consideration. It is true that Gov. Spotswood sometimes speaks of four furnaces; but all the facts go to prove that he was interested in only the first and second named above, and in one "air-furnace," located at the outlet of the Massaponax river, on a neck of land now owned by Dr. Morton. This Massaponax furnace was built by the master-workman who came from England expressly to erect the blast-furnace at Fredericksville, and is known to have been a reverberatory melting-furnace of the long-hearth type so common in Europe at that period; the capacity being about four tons daily output of cast-iron. In later records, mention is made of another air-furnace, which was counted as belonging to this section of the country, although, being located on the Maryland side of the Potomac, it was out of the colony of Virginia.

The Rappahannock furnace was erected, without doubt, on the arrival of the Palatines at Germanna. Its location, in proximity to the ore-beds, will be clearly understood by reference to the accompanying sketch-map, which shows also its position relative to Germanna, as Gov. Spotswood's main headquarters, and to the air-furnace at Massaponax. This Rappahannock furnace stood close to the fords of the river, and on the almost precipitous banks of a small stream which was relied on for the necessary power. Considerable grading was done to provide for the operation of the furnace, the storage of charcoal, limestone, ores, etc. Protected by its isolated position, the structures remained in an excellent state of preservation. The furnace is almost intact, its cleanly cut stone-work and auxiliary walls, race and chambers being about as they were left by the last workmen, 160 years ago. A giant black-walnut tree is growing from the mouth of the furnace, binding with its roots the heavy walls in place, and closing every aperture against the ravages of time. The slag-piles, fluxes, coal and considerable amounts of pig iron, found in odd shapes in the cinder-beds and the bed of the brook; the dam of generous extent but faulty construction; the

race-way, tail-race, etc., are interesting mute evidences of this early enterprise.

The ore-banks, close at hand, are extensive and might well excite the remarks of Col. Byrd, quoted above.

Only the mines nearest this furnace were used. There are old banks about the Catherine furnace, in that region; but these have been opened since 1830, to supply that particular plant.

The second furnace was located, as will be seen on the map, in an isolated position, off the main lines of travel, about thirty miles from Fredericksburg, ten miles from the present Mineral City, one-half mile from the North Anna River, and about twenty miles due south of Germanna. The exact site is just off the main county-road from Fredericksburg to Mineral City. It has been very interesting to discover the reasons for the choice of such a place, so far from Germanna, and attended with such drawbacks as the high cost of transportation for building materials and for pig-iron (marketed, perhaps, hundreds of miles away), and the peril to the lives of the workmen at this exposed outpost. But an examination of the whole mineral-belt shows that no other site on the lower end of it would have been equally suitable. Geographically, it is about in the center of the belt, as then already well-known, near the North Anna river (which at that date must have been a stream of considerable importance), and in close proximity to the best of brown iron-ores, which are scattered throughout the primitive rocks of the region. Alongside this outcrop of pure ores, there lies a wide belt of garnetiferous, micaceous and talcose slates. The furnace was placed just at the dividing-line between ores and slates, in a gash-seam which supplied sufficient water for all purposes and which crossed the slates at right angles, opening to the southward toward the river. The dam, built about 1000 feet above; the race which brought the water down to the furnace, at a level probably 20 feet above the base; the walls of the flume which carried the water to the overshot-wheel, and the site of the blowing-apparatus (probably the accordeon type of bellows), are all still to be seen on the spot. Just above the furnace are levelled spaces for the storage of ores, charcoal, fluxes, etc., sufficient for at least twenty thousand tons of materials. Still above this level, the ground rises to a height of probably 60 feet, and the ore-pits, with samples of the ores used, are easily found for some distance to the southwest.

The casting-beds were nearly on the level of the brook, about 50

feet away; and the furnace must have been at least 20 feet high, from its base to the bridge which extended to the bank above.

The slag-heaps are some distance away, and contain at least 10,000 tons, possibly double that amount, as they have been in part washed into the brook, overgrown and buried, and it is therefore difficult to estimate their contents closely.

The materials which I have gathered, prove that the furnace actually smelted iron-ores and produced good pig-iron; the slags plainly indicate excellent judgment in furnace-management; and the quality of the pig-iron proves the knowledge of ore-mixtures to have been equal to that of the present day in the same district. Ores, fluxes, charcoal, masonry, etc., abound in the great quantity of *débris* scattered about. Unfortunately, no care has been taken by the owners of the land during the last twenty years for the preservation of the stack itself, or its materials; and at least half its original height has been sacrificed to furnish foundations and chimneys for the cabins of the neighborhood. A few of the bricks (unique in their shape and weight and evidently brought from some foreign land), are still in the ruins of the furnace; but the choice bricks of the upper part have disappeared. The specimens I have secured were taken some time ago, during a further wrecking of the main structure, and represent a fair average of what now remains, perhaps 10 feet in height. These samples represent the pudding-stone, which made up the body of the furnace, with streaks of slag filling all the joints and seams; the fire-stones, both sandstone and slate; and the inner and outer bricks, the mortar on which is as firm as when first seasoned, although it has been exposed to weather for 170 years or more.

Col. Byrd in *A Progress to the Mines*, page 49, says of these works:

"I passed the south branch of Pomunky river. . . . After this I had eight miles to Mr. Chiswell's. . . . I found Mr. Chiswell a sensible, well-bred man, and very frank in communicating his knowledge in the mystery of making iron, wherein he has had long experience. I told him I was come to spy the land, and inform myself of the expense of carrying on an iron work with effect. That I sought my instruction from him, who understood the whole mystery, having gained full experience in every part of it; only I was very sorry he had bought that experience so very dear.

"He answered that he would, with great sincerity, let me into the little knowledge he had, and so we immediately entered upon the business. He assured me the first step I was to take was to acquaint myself fully with the quantity and quality of my oar. For that reason I ought to keep a good pick-axe man at work a whole

year, to search if there be a sufficient quantity, without which it would be a very rash undertaking. That I should also have a skillful person to try the richness of the oar. Nor is it great advantage to have it exceeding rich, because it then will yield brittle iron, which is not valuable. But the way to have it tough is to mix poor oar and rich together, which makes the poorer sort extremely necessary for the production of the best iron. . . . He told me, after I was certain my oar was good and plentiful enough, my next inquiry ought to be, how far it lies from a stream proper to build a furnace upon, and again, what distance that furnace will be from water carriage; because the charge of carting a great way is very heavy, and eats out a great part of the profit. That this was the misfortune of the mines of Fredericksville, where they were obliged to cart the oar a mile to the furnace, and after 'twas run into iron, to carry that 24 miles, over an uneven road to Rappahannock river, about a mile below Fredericksburg, to a plantation the company rented of Col. Page. . . . He told me farther that 120 slaves, including women, were necessary to carry on the business of an iron work, and the more Virginians among them the better; tho' in that number he comprehended carters, colliers, and those that planted corn. If there should be much carting, it would require 1600 barrels (8000 bushels) of corn yearly to support the people and the cattle employed; nor does that quantity suffice at Fredericksville. That if all these circumstances should happily concur, and you could procure honest colliers and firemen, which will be difficult to do, you may easily run 800 tons of sow-iron a year. The whole charge of freight, custom, commissions, and other expenses in England, will not exceed 30 shillings a ton, and 'twill commonly sell for £6, and then the clear profit will amount to £4 10s. So that, allowing the 10 shillings for accidents, you may reasonably expect a clear profit of £4, which being multiplied by 800, will amount to £3200 a year to pay you for your land and negroes. But then it behooved me to be fully informed of the whole matter myself, to prevent being imposed upon; and if any offered to put tricks upon me, to punish them as they deserve.

"In the afternoon I walked with my friend to his mill, which is half a mile from his house. It is built upon a rock very firmly, so that 'tis more apt to suffer by too little water (the run not being over plentiful), than too much. On the other side of this stream lie several of Colonel Jones' plantations. The poor negroes upon them are a kind of Adamites, very scantily supplied with cloaths. Close by the furnace stood a very spacious house full of charcoal, holding at least 400 loads, which will be burnt out in three months. The fire in the furnace is blown by two mighty pair of bellows, that cost £100 each, and these bellows are moved by a great wheel of 26-foot diameter. The wheel is carried round by a small stream of water, conveyed about 350 yards over land in a trough from a pond made by a wooden dam. Chiswell, the only person that knows anything of the matter, has £100 a year, and richly deserves it.

"At our return, we saw near the furnace large heaps of mine, with charcoal mixed with it, a stratum of each alternately. To this they put fire, which in a little time spreads through the whole heap and calcines the oar, which afterwards crumbles into small pieces fit for the furnace. There was likewise a mighty quantity of limestone brought from Bristol by way of ballast, at 2/6 a ton, which they are at the trouble to cart hither from the Rappahannock river, but contrive to do it when the carts return from carrying of iron. They put this into the furnace with the iron-oar, in the proportion of one ton of stone to ten of oar, with design to absorb the sulphur out of the iron, which would otherwise make it brittle. And if that be the use of it, oyster shells would certainly do as well as limestone, being altogether as strong an alkali, if not stronger.

"But the founders will by no means be persuaded to go out of their way, though the reason of the thing be never so evident. I observed the richer sort of mine, being of a dark color mixed with rust, was laid in a heap by itself, and so was the poor, which was a liver or brick color.

"The sow-iron is in the figure of a half-round, about two feet and a half long, weighing 60 or 70 pounds, whereof 3000 weight make a cart load drawn by eight oxen, which are commonly shod to save their hoofs. When the furnace blows, it runs about 20 tons of iron a week.

"I took my leave about ten and drove over a spacious level road ten miles to a bridge built over the river Po. Two miles beyond that we passed by a plantation belonging to the company, of about 500 acres, where they kept a great number of oxen. Three miles further we came to the Germanna road, then eight miles to the main county road that leads from Fredericksburg to Germanna, which last place I reached in ten miles more."

I have already quoted Colonel Byrd's remarks upon his visit to the Rappahannock furnace, which he left by the main road, over which all the cast-iron was carted to the "air-furnace" at Massaponax, five miles below Fredericksburg.

Another quotation from him, covering the information which he derived from dwellers in Fredericksburg, concerning the Washington furnace, may conclude this portion of the history:

"On the other side the river, in King George county, twelve miles from hence, is a spring of strong steel water, as good as that at Tunbridge wells. Not far from this spring are England's iron mines, called so from the chief manager of them though the land belongs to Mr. Washington. These mines are two miles from the furnace, and Mr. Washington raises the oar and carts it thither for 20/ the ton of iron that it yields. The furnace is built on a run, which discharges its waters into Potomeck. And when the iron is cast, they cart it about six miles to a landing on that river. Besides Mr Washington and Mr. England, there are several other persons concerned in these works. Matters are very well managed, and no expense is spared to make them profitable, which is not the case in the works I have already mentioned. Mr. England can neither write nor read, but without those helps is so well skilled in iron works that he don't only carry on this furnace, but has likewise the chief management of the works at Principia, at the head of the bay, where they have also erected a forge and make very good bar-iron."

These interesting proofs of successful iron-smelting at so early a period in the history of the colonies must inure principally to the individual credit of Governor Spotswood, in whom foresight, energy and perseverance were predominant traits.

Among the difficulties which he had to surmount by his personal management, were, for instance, the handling of a colony of dissatisfied foreign workmen, who, allured to America by the promises of De Graffenreidt, found themselves thrown into a wilderness almost unknown and full of dangers; the laying out of roads and the bringing

in of supplies; the prospecting for minerals in the yet more remote wilderness; the locating of homes and opening up of agricultural tracts to support increasing immigration; the erection of an entirely new class of furnaces, and the working of these furnaces and the sale of their products, with all the vexing details which are partially made known in the records here presented. What he accomplished in this new field, and under circumstances which seem to us to have been almost impossible; the attention which he attracted, and the reputation which he made among the leading iron-masters of the old world, by the introduction of the "Plantations Pig-Iron" into their markets; and the assistance he rendered to all struggling industries, while thus laying the foundation of one of the most important of them all, constitute a title to honorable fame, in the presence of which criticism of his personal ambitions may well find itself disarmed.

In the chain of historic evidence, which secures to Virginia the credit of establishing the first bloomery in the year 1619, and records for her an almost unbroken line of successes in charcoal furnace-practice since the Revolutionary War, there has been hitherto (by reason of the unfortunate destruction of documents) one missing link, namely, the proof of the extensive pioneer work done by the Spotswood furnaces from 1714 to 1730. This paper describes how I have sought to recover that link, establishing beyond doubt, by careful tracings along the mineral belt from the South Anna to the Rappahannock, the sites of the first and second furnaces located and worked by the German Palatines and their successors. It is clear that this colony was settled at Germanna to open the mines and prepare for the active work of the furnaces; that materials of various kinds, which we now find on the ground, were brought from abroad for the construction of the first furnace at Rappahannock, and for the second furnace at Fredericksville; and that we have here the first pig-iron ever smelted in America from the ore, in a furnace in which forced blast was used.

Discussion.

Mr. E. C. Pechin, Roanoke, Va.: I have been very much interested in Mr. Adams's paper, and I think he deserves a great deal of credit for having so carefully worked out this inquiry. Those of us who have anything to do with mining in Virginia know one thing to our sorrow, that we do not have to go back to 1600 to find uncertain titles or defective records and boundaries in

a great many parts of that State and of West Virginia. Mr. Adams has shown much skill in clearing up the obscurity due to such causes, surrounding his subject.

The early document from which he quotes, and which says that "all the land hereabouts seems paved with iron-ore, so that there seems enough to feed a furnace for many ages," sounds familiar to a reader of Southern literature of the last one or two years. Indeed it is a comparatively mild statement!

But, seriously, it is a point of very great interest that these original iron works were established, as Mr. Adams shows, for the purpose of working the "gossan"-ore, which forms the oxidized cap, above the water-level, of the deposits of iron sulphides carrying copper-ores. The sulphur and copper have almost disappeared, and the gossan is practically a lean hydrated iron-ore. The sulphur is very low indeed. The copper varies from a quarter of one per cent to one per cent. As you approach the sulphides underneath this capping of iron-ore you get a zone of so-called "black oxide" of copper, and years ago the whole belt was extensively worked by copper-producers before the great copper-mines of Lake Superior came in. This black oxide of copper lies right under the gossan-ore and next to the mundic rock, and then the latter continues down to unknown depths. Mr. Adams, at Mineral City, has a shaft 500 feet deep.

This gossan-ore has been largely ignored as a material for the manufacture of iron. Miners turned to the brown ores of the Blue Ridge. Over and over again, in travelling through the mountains of Virginia, I have come upon the ruins of old furnaces in secluded valleys, miles and miles away from any railroad, which had once smelted the brown ores; yet nobody living in the locality recollected having seen them worked, or had any knowledge of them beyond vague traditions. I recollect one old stack on Irish Creek that is called the "Colonial Furnace," which was probably built before the Revolutionary War. An old half-Indian half-negro that I talked with told me that his father used to tell him about having worked at this furnace. I suppose they made half a ton a day there.

Most of these old furnaces seem to have run entirely on the brown ores; but there are in Floyd county the remains of an old furnace which ran entirely on this gossan-ore, and there is a very curious story about it, to this effect: The proprietor of this little furnace made one or two tons a day, which he would run into castings, kettles and pots for the use of the mountaineers around there. On one occasion, before he had found a market for his wares,

he took a lot of them over to a county fair, and vainly tried to sell them. They were rather a rough-looking lot of utensils, and did not attract favorable attention, and the old man began to feel discouraged, and did not think he would be able to sell any. Some one was jeering him about his bad luck, and in a fit of rage he took up one of these pots and said he would break the "danged thing to pieces," and he smashed it down on a big rock. To the amazement of the beholders and himself, instead of breaking, it rebounded several feet in the air, which created such a sensation that the old man immediately doubled his prices, and in the course of an hour had entirely disposed of all the goods he had brought with him. They have in the South the record of car-wheels made out of this sort of iron, before the war, that are reported to have far exceeded the record of any other car-wheel either in this country or abroad. The only explanation which I have heard of this is the extraordinary tenacity given to the iron by reason of the small percentage of copper present. At the Pulaski furnace to-day they are making one of the most beautiful foundry-irons that I have ever seen anywhere. It is "as soft as lead," and as tough and strong as anything I have ever seen. They get that quality of iron when they mix say thirty per cent. of this gossan with their brown ores. The brown ores of the region run high in phosphorus and manganese, but by mixing thirty per cent. of this with them, iron is obtained for which there is a very large demand.

The gossan-ore is not very rich in iron. Prospectors, of course, pick their samples and get a high percentage of iron, and so it was originally assumed that this ore was going to yield fifty per cent. When I made my first examination of the property, eighteen months ago, I came to the conclusion that with proper care in mining they could probably calculate upon forty-five per cent. As the mines go now they are getting about forty-two. The following analysis was made of a sample taken from eight cars picked at random from the shipments within the last two or three weeks :

	Per cent.
Iron,	41.74
Silica,	19.
Manganese,	0.13
Phosphorus,	0.13
Sulphate,	0.28
Aluminum,	2.68
Lime,	0.96
Magnesia,	3.46
Potash,	0.16
Soda,	1.79

Combined water not stated.

Of course this is, strictly speaking, a brown hematite; but it will be readily seen that the alumina and magnesia, soda and potash in this somewhat unusual mixture are going to help very largely in the treatment of the ore in the furnace, by helping to get rid of the silica.

MR. BIRKINBINE: Was that sample dried at 212°; and if so, how much water did it lose?

MR. PECHIN: Yes; and it may have lost from 6 per cent. up to 500 per cent., if you will permit an exaggeration! It is a spongy, open, porous ore; and if you both mine it and get it to a furnace on a dry day it will arrive with perhaps 5 to 6 per cent. of moisture. But if it were caught in a little gentle shower, such as we have just had, *in transitu*, it would reach the furnace absolutely soaked with water. You may put a lump of it into a bucket of water, and you will soon be surprised to find how little water remains in the bucket, while the lump is there just the same. I am inclined to think that when mining operations shall have been opened up on a somewhat larger scale, so that the work can be regulated, the true policy of the furnaces using that ore will be to get their supplies during the summer and put it under cover, and not work the mines at all during the winter or rainy season. I doubt whether the gossan-ore will ever be smelted by itself, although parties interested, who have been investigating the admirable qualities of this iron, made with charcoal in olden times, are now seriously entertaining the proposition of building a charcoal furnace close to the mines, to make car-wheel iron. For malleable castings I can imagine nothing better.

It is curious that, after this ore has been ignored for 150 years, (having been, according to Mr. Adams's paper, the supply of the original Virginia furnaces), it should only have attracted attention within the last year or two, and that it is now coming before the public. To understand exactly what this means to us in the South, it must be remembered (as I have pointed out in one of my papers in a general way,) that our brown ores are highly impregnated with phosphorus and manganese, and it is difficult for us to get what may be called an extra-strong foundry-iron. Our forge-irons are very good, owing to their extremely low silicon. This gossan-ore seems to have the effect of giving strength to the brown ores, and it looks very much as if it were going to play a very important part. Careful experiments are on foot to determine whether the sulphur cannot by some cheap process be extracted from the mundic rock on a commercial scale. If that should be successfully done, the quan-

tity of the available iron-ore (including the desulphurized residuum) will be indefinitely large, for the great sulphur-lode can be traced for miles. I have myself followed it for twenty miles right through the country, and it varies from two feet to many feet in width. You can see, therefore, that there is an enormous supply if it can be made commercially available.

R. W. RAYMOND, New York City: I suppose this deposit is the same as that of the Ducktown copper-mines in Tennessee, which was traced into Virginia twenty odd years ago, and worked there for copper at the Betty Baker and other mines. Digging away the gossan, the miners found along the underground water-line what had leached down and deposited itself in the so-called "black oxide" copper-ores. These are easily reduced, and, while they last, furnish a material which can be reduced to black copper at one smelting. If I am not mistaken, the usefulness of the gossan outcrop as a source for the manufacture of iron has not been so completely ignored as Mr. Pechin supposes. Some twenty-five years ago, in connection with the editing of a report by Dr. Hermann Credner, on the Ducktown mines, my attention was called to the value of these outcrops as iron-ore; and I was then informed that the Tredegar iron-works, of Richmond, had made, before and during the war, from this kind of ore, iron of a very high quality, a good deal of which went into the artillery and small arms manufactured for the Confederate States.

I might also say that the crystalline schists of more Northern States show a similar zone or zones of heavy pyritic beds. I believe they are found in Pennsylvania, and I have examined them at Anthony's Nose in the Hudson Highlands, and at Litchfield, Conn. The pyrites is not the yellow mundic; it is the so-called "pyrrhotite" or brown magnetic sulphide. It occurs again at Sudbury, Canada, over 150 feet in thickness. It is not generally covered with heavy gossan; but it seems to carry everywhere enclosed chalcopyrite and also nickel. The percentages of copper and nickel are variable. At Anthony's Nose both copper and nickel are low, and the same is true at Litchfield, Conn. I think Mr. Wharton's old nickel-mine in Pennsylvania is in a similar formation, and I believe that nickel as well as copper has been found in the zone of the Ducktown mines. If I am right, we have here rather a great geological formation than a single ore-deposit, as we are accustomed to conceive such deposits. There is no doubt that whatever may be its wealth or its poverty at any given locality, the aggregate of

the product of this belt or system of pyritic deposits will be immense, and whether we at this time do or do not utilize it wherever it occurs, our children will not be entirely without resources for iron-ore in the future, even when the original ores of their fathers are exhausted. I would say with regard to the peculiar qualities which are attributed to this ore that I cannot myself understand that they have really any particular connection with its porous condition or with its origin as the result of the oxidation of the magnetic pyrites. If they have, then "blue-billy" ought to have the same qualities, and, for all I know, it has them. The process that has gone on along this great outcrop of Virginia has undoubtedly removed all soluble impurities, and I doubt whether there ever was much phosphorus. My recollection is that in the manufacture of sulphuric acid at Staten Island, twenty years ago, the material proved singularly free from phosphorus, though not from arsenic. We also know that titanic acid is not present. Thus we have none of the impurities that would affect foundry-iron, with the exception of sulphur. If we can with sufficient cheapness remove the sulphur, we may get one of the purest raw materials, apart from the earthy gangue, of which there is, in many parts of these massive deposits, but a small percentage.

A REVIEW OF THE IRON-MINING INDUSTRY OF NEW JERSEY.

BY JOHN C. SMOCK, TRENTON, N. J.

(Cleveland Meeting, June, 1891.)

THE rich deposits of magnetic iron-ore in the Highlands of northern New Jersey attracted the attention of iron-workers at the time of the earliest settlements in that region. The outcrops of the ore-shoots at the Dickerson mine, at Mount Hope, and at Hibernia, in Morris county, and at Ringwood, in Passaic county, which were doubtless such as would lead to discovery, location and development of nearly all the large iron-mines in that part of the State, appear to have had their beginnings where the ore cropped out on the surface or where it was scarcely concealed by a thin mantle of earth. In the early history of New Jersey iron-mining, the careful explorations and surveys made by the agents of the famous London Com-

pany, organized by Peter Hasenclever, resulted in the discovery of nearly all the large mines, which are still worked, in the north-eastern part of Morris, and in Passaic county. The explanation is not, as has been said, that there are no more shoots or beds of ore to be found, and, consequently, no new mines to be opened; but that the localities of outcrops, where the ore-bodies came to the surface, have been mostly found. It is highly probable that there are few, if any, places as yet undiscovered in this region, where iron-ore is to be seen on the surface, or, as indicated by strong magnetic attraction, is but lightly covered. The early prospectors enjoyed the advantages belonging to pioneers in a virgin territory, and reaped a rich harvest in the discovery of mines, the long-continued productiveness of which has overshadowed the developments of a century and a half of succeeding workers in that field.

The mining of iron-ore in the Highlands may be said to have begun at the Dickerson mine about 1710.* The development of the industry was rapid, and the manufacture of iron in forges in that district was started at about the same time. At first the product was for home consumption, and the ore was used in the forges within a few miles of where it was mined. It was a day of small beginnings, and the forges were run by water-power on many sites, where at the present time available power is scarcely recognized. The small demands made but slight impression upon the large ore-shoots, and mine exhaustion was an apparent impossibility at the rate of production of that early period. The era of forges was succeeded by that of charcoal blast-furnaces. Acrelius, the historian of New Sweden, who lived in this country from 1750 to 1756, mentions five iron-works, one of which, that at Oxford, was built in 1742. The others were the furnace at Ogden's (now Ogdensburg), the Union Iron Works, the Mount Holly furnace and the Stirling furnace, near the New York State line.† The furnaces at Andover, Ringwood, Mount Hope and Charlottenburg, were built about 1763–1772. These furnaces consumed much more ore than the small forges which preceded them. The iron manufacturing industry was in a prosperous and growing condition up to the period of the Revolution. There are no available data from which to ascertain the amount of ore used annually, other than the estimated capacity of the furnaces named. Using this basis it is safe to say that the

* The iron works at Tinton Falls, in Monmouth county, built about 1674, smelted the bog-ore found in that county.

† Swank's *Iron in All Ages*, Phila., 1884, pp. 117, 118.

annual output of the mines worked at that period did not exceed 10,000 tons. Morse's *American Gazetteer* (Boston, 1797) says that in Morris county alone, "there are no less than seven rich iron-mines, from which might be taken ore sufficient to supply the United States." Some forges and furnaces of the ante-revolutionary period used bog-ores, found in the southern part of the State; and they continued to work on these ores well into the present century. In 1802 there were, according to Swank's history of *Iron in All Ages*, thirteen furnaces and one hundred and fifty forges in New Jersey. Gordon's *Gazetteer* puts the total output of the iron-mines in 1830 at 20,000 tons. Statistics of production for the decade 1830–40 are wanting, and the progress of iron-mining as measured by the output is unknown. Prof. Henry D. Rogers, State Geologist, gives descriptive notes of the important mines and mining districts, as he saw them at the time of his survey, the final report of which was published in 1840. He has notes on the Ringwood, Hibernia, Mount Hope, Succasunna and other mines in Morris county. The "relative fewness" of mines in the Wallkill, Schooley's and Musconetcong Mountains, is referred to as notable; and prospecting for ore in these localities is said to be promising. To the southwest, the Oxford Furnace ore-deposits are described as large and containing an "enormously great quantity of ore." Prof. Rogers refers to the hot-blast and the use of anthracite coal as "new methods recently introduced," and about to lead to a brighter era.

In 1855 Dr. William Kitchell, State Geologist, estimated the output for that year to be 100,000 tons. Many of the mines described in his report are still productive, and few new mines have been opened since that date in Passaic, Sussex and Morris counties. About eighty were mentioned in his list for 1855. In his report for 1856 he says, "they could be made to yield advantageously no less than one million tons of ore annually for many years to come, which would be sufficient to supply half of the present annual consumption of iron in the United States."* At that time there were three furnaces in New Jersey which used anthracite coal as fuel in smelting iron-ore. The larger part of the ore then mined in the State was used in Pennsylvania. In his second report he says, "there is no reason then, why New Jersey should not become, *par excellence*, the greatest iron-producing State in the Union"—a hope unrealized.

* *Third Annual Report on the Geological Survey of the State of New Jersey*, for the year 1856, Trenton, 1857, p. 10.

There are no accurate statements of the amount of ore mined yearly in the State after the suspension of the Geological Survey until 1860, when the United States Census reported a total production of 164,900 tons. In 1863 the estimated output, according to Prof. Cook's report for that year, was at least 200,000 tons, the Morris Canal Co. having carried 176,531 tons of ore. For the next year the estimate was 226,000 tons. The increase in the production was gradual and steady throughout the decade to 1870. The *Geology of New Jersey*, published in 1868, gives a full list of the iron-ore localities, producing as well as abandoned mines. There were at that time one hundred and fifteen separate mines and mine-groups, and their aggregate output was 275,000 tons. The list had increased to one hundred and sixty-one in 1871. Nearly all of the additional locations were in Warren and in Sussex counties, whereas in the northeastern part of the district, in Morris and in Passaic counties, there were few discoveries outside of the large and well known mine-groups, and these were merely new shoots of ore in active and working mines of long history. About thirty mines and mine-groups furnished the greater part of the product of the State at that period—1868–1871. Many of the remaining one hundred and thirty-one localities of the list were either new openings as yet undeveloped, or abandoned mines.

There are, beginning with 1870, yearly returns of the iron-ore production, which approximate closely to an accurate census of the mines. The accompanying diagram shows the fluctuations in the production of the whole State for the period 1870–1890, inclusive. The upper line represents the movement in the yearly output; the lower one shows the price of anthracite pig-iron at Philadelphia year by year for the same period. The rapid upward movement in 1872 and 1873 represents the buoyancy of the iron market and the influence of high prices, continued into 1873, and beyond the maximum price of iron in 1872. The advance in the output of 1871 upon that of 1870, and of 1872 upon that of 1871 was at the rate of 25 per cent. per annum. The increase was in the greater output of the larger mines, and in the number of producing localities, including some mines which had been idle since the high prices at the close of the civil war. The Hibernia mines reached an aggregate of 90,000 tons of ore. The completion of the Midland Railroad afforded a new outlet to the northern part of the district, which stimulated production and led to the opening of several new mines. There was a great deal of searching in Warren county,

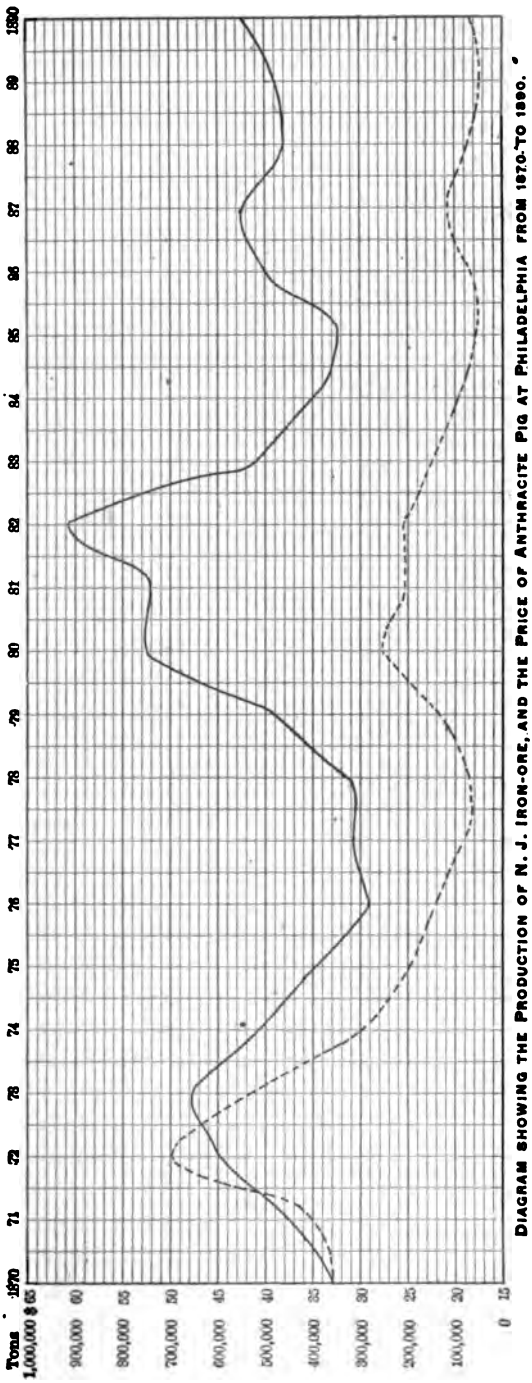
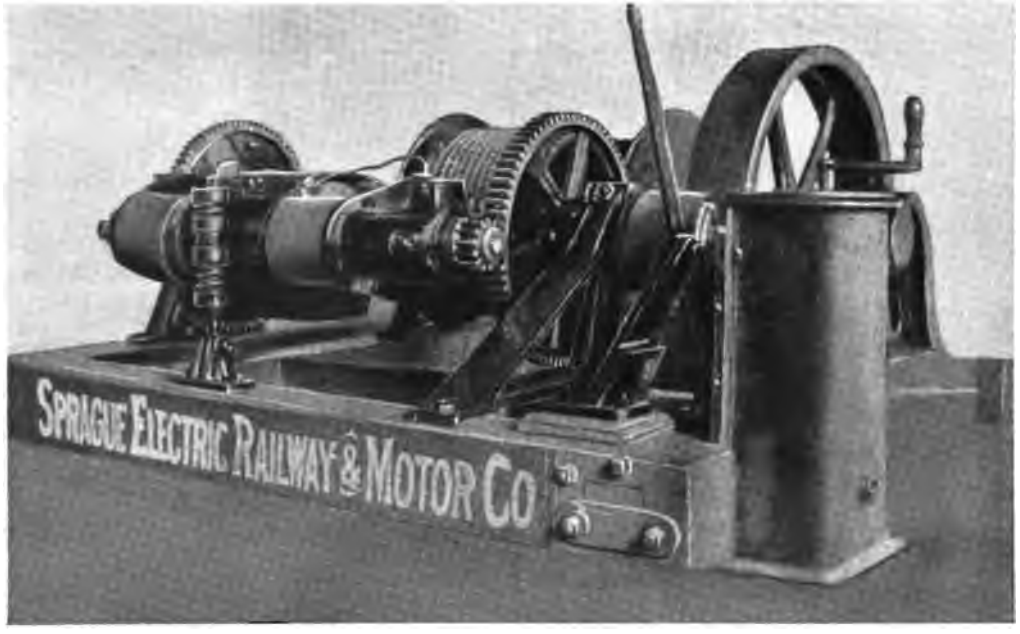


DIAGRAM SHOWING THE PRODUCTION OF N. J. IRON-ORE, AND THE PRICE OF ANTHRACITE PIG AT PHILADELPHIA FROM 1870 TO 1890.

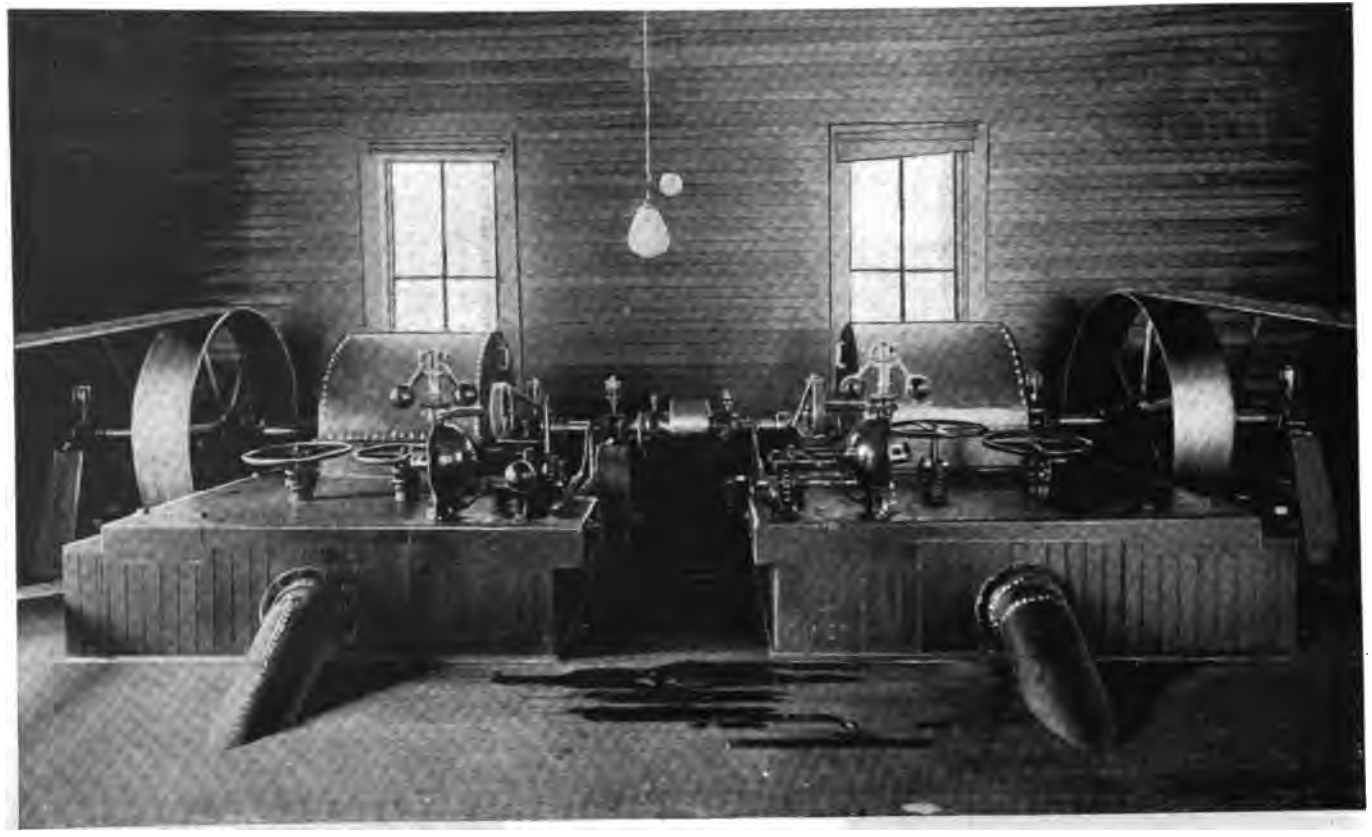
also, and new localities of ore were discovered, but almost without exception they were mines which could not be profitably and continuously worked. The Geological Survey made another full list of iron-mines in its report for 1873, and represented the condition of the industry at the epoch of good prices and of the maximum production. The aggregate output was 665,000 tons, which was not surpassed until 1880. The number of mines in the list was two hundred, but the majority were localities in the various stages of development from exploitation to producing mines, few of which could be properly designated as iron-mines. The descriptive notes of that report show many mines of lean ores which are to-day idle because of their low metallic percentage. The average valuation of the ore at five dollars per ton also shows the stimulus to mining lean ores and working thin beds and small shoots. As in 1868, the larger part of the total production was from the few large mines. The decline in prices made itself felt in 1874, and the production was 20 per cent. less in that year than it was in 1873, although new openings were made and mining was carried forward with vigor at many of the new mines, in the hope of a recovery in price and a renewed demand for ore. These expectations were not realized, and under the weight of depressing prices the State's production declined to 390,000 tons in 1875, and touched bottom at 285,000 tons in 1876, or nearly the mark for 1867. Several of the large producers suspended work, and all of the smaller and weaker companies and mines on lean ores were compelled to stop altogether. Thirty of the two hundred in the list of 1874 were able to continue active through the panic for the years 1873–1879. The rich ores and those suited to Bessemer steel found a market at barely remunerative prices. The boom of 1879, following the general business revival, is evident in the sharp upward rise in the curve during the years 1878–1880. The activity in mining was as great in the latter part of 1879 as it had been in 1872–3, and work was resumed at many of the old mines which had been idle since the panic of 1873. The high prices and the demand occasioned a widespread inquiry for ore, and there was a return of miners and capital to the district comparatively more rapid than the exodus at the beginning of this period of depression. A new era of prospecting and of developing properties was witnessed. As a result the output for 1879 was nearly twice as great as that of 1876. Under this stimulus the list of localities contained three hundred mines and open-

FIG. 2.



Original Sprague Motor used by the Aspen Company.

FIG. 5.

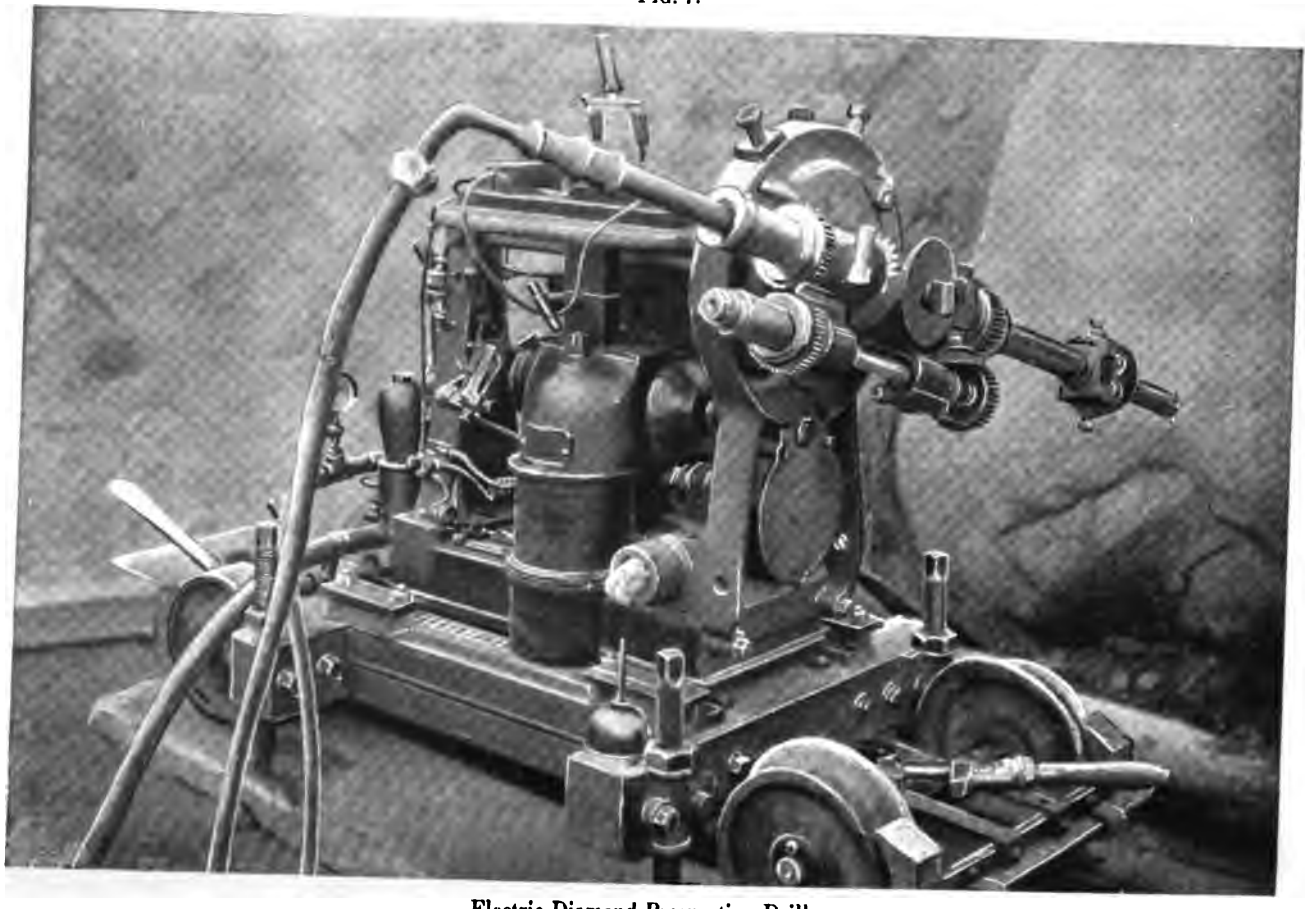


Water-wheel, Shafting, Governors etc.

FIG. 6.

Water-wheel and Dynamo.

FIG. 7.



Electric Diamond Prospecting-Drill.

ings, or one hundred more than that of 1873.* The high price of iron continued into 1880, and the active demand for ore resulted in an aggregate production in that year of 745,000 tons, an advance of over 50 per cent. upon that of 1879. In nearly every locality where ore was known to occur, openings were made without careful exploitation, and regardless of cost of mining and conditions favoring permanent productiveness. The tenth census of the United States gave the statistics of the census year as 757,372 tons of iron-ore, and the capacity of the mining establishments at 1,487,829 tons. The rank of the State was fourth in the amount of iron-ore raised, the same as it was in 1870. The influence of good and steady rates for iron and iron-ore throughout 1881 and 1882 was favorable to an increase, and the output reached the climax in 1882, amounting to 932,726 tons. The upward curve from 1878 to 1882, and the sharp decline for 1883–1885 are notable. The output in 1883 was 44 per cent. below that of 1882. Work was suspended at many of the larger mines, and the ill-equipped and smaller ones were abandoned. Searching for new ore-localities was at a standstill. The decline in the cost of labor was less than that in the price of ore, and the royalties were generally as high as in the preceding year. Under these depressing conditions the industry suffered a further decline, which was continued through 1884 and 1885. The importation of ores for Bessemer pig-iron met the demand of the eastern manufacturers in part, and helped to cause a still more serious condition in the business and a decreased production, until the minimum was reached in 1885, in which year 330,000 tons of ore were raised, a limit nearly as low as that of 1876.

Since 1885 there is an upward curve, representing the State's production, with a slight falling off in 1888—this downward tendency being less marked than it was in 1879–80. The period has thus been one of more steady prosperity, although lacking the phenomenal activity of what might be termed a "boom" in iron-mining.

Reviewing the changes of the past two decades, it is a prominent fact that only a small number of the new openings have been developed into producing mines, and of this number two only have been worked more or less continuously—the Kishpaugh, dating from 1871, and the Queen, from 1882. Other mines, opened since 1870, which have yielded ore in quantity, but have not been steadily worked, are: Langdon, Gulick, Hedges, Swayze, and Squier's (near Chester),

* This list has the groups of mines resolved into individual and separate mines.

Green Pond (new openings), Stoutenburgh, on Schooley's Mountain, Gove, near Drakesville, in Morris county; Lanning, at Oxford Furnace, and Pequest, in Warren county; and Board and Centennial or Squire's mines, in Passaic county. Of the older and once well-known mines, the Andover has not been worked in more than thirty years; the Dalrymple, Byram, Randall Hill, Baker, and others have also been abandoned, apparently without hope of resumption. There are others which have been idle since the "boom" of 1879–82, and are in all probability destined to remain so. The list in the report of the Geological Survey for 1883 includes the names of three hundred and twenty-five mines. Of these, thirty only are to-day being worked.

The extent of the workings and their great depth are also notable in the case of the larger mines of the State. The Hurd mine, at Hurdtown, is down 3625 feet on a slope of 30°; the Dickerson is 1200 feet deep; the Mount Pleasant 900 feet, and the Hibernia 800 feet. The increase in depth has been at a rapid rate. The Dickerson has been deepened 500 feet in seven years; the Hurdtown mine slope has been lengthened 1600 feet since 1883; the Hibernia mines have opened an ore-bed more than a mile long; the Mount Hope line is nearly as long, and the Richards, Teabo, Mount Pleasant, Irondale and Ogden mines are examples of extensive opening, both in length and in depth. The rapid extension of the workings in these old mines has been caused by the growing demand for ore, and the increased aggregate of production for the State is due to their greater activity and larger output rather than to any increase from new localities and more recently developed mines. The result has been a rapid exhaustion of the higher and more accessible shoots, necessitating deeper mining following the descending shoots of ore. Notwithstanding this increased depth, the average output per mine is much greater than it was two decades ago. During the past few years the mine royalty has been reduced, which has afforded a practicable margin for profitable working, and helped to maintain the prosperity of the district, enabling it to hold its position as an iron-ore producer among the eastern United States. New Jersey has not advanced by such rapid strides as the Lake Superior districts, or as the Southwest; but it has kept pace with New York, Pennsylvania, Ohio and Missouri in the production of iron-ore. Its present rank is sixth, having receded two points since 1880.

In this retrospect it may be noticed that the sanguine predictions of iron-masters and of geologists of the earlier or mid-century years

have not been fulfilled; and the State has not contributed its proportionate share of the total amount of iron-ore consumed in the country, as compared with the past. The growth has been slow as compared with the almost marvellous developments in the Marquette, Menominee, Gogebic and Vermilion districts. The total consumption of iron-ore in the country has increased tenfold during the last thirty-five years; the State's production, however, taking the average of the last five years, was only five times what it was in 1856. It must be observed that a large part of this advance was previous to 1870. The average production per annum for 1870–79 is 449,034 tons; that for 1880–89 is 533,824 tons, equivalent to a gain of twenty-five per cent. The total output for the twenty-year period, 1870–1889, inclusive, amounts to 10,128,575 tons. Dr. Kitchell estimated the output of the New Jersey iron mines in 1856 to be seven per cent. of all of the domestic ore used in the country; in 1890 the State furnished a little less than three and a half per cent. of the ore for home consumption.

The comparison of the statistics of New Jersey and of the whole country, and review of the history of the industry in the State, as indicated by the rate of production by years and by terms of years, lead to the following conclusions:

1. The iron-mining industry of New Jersey began in the colonial period, and grew slowly with the early settlement of the district.

2. The early development was at many points, and there are comparatively few productive localities which do not antedate the beginning of the present century.

3. The rate of increase in the production was slow up to the epoch of the hot-blast and the anthracite coal furnace, about 1840.

4. Since 1860 it has been characterized by sudden and wide fluctuation, influenced by the price of iron and the demand for ore.

5. The advance in the production of the State has been much slower than in that of the whole country.

6. The steady progress in the direction of more economic working and the average increase of product, as seen in comparing decades of production, show that the capacity of the mines in the aggregate is adequate to a still greater output, and their enduring prosperity is such as to give hope of a future advance.

7. The stage of slower growth has been entered upon by this industry in the State of New Jersey, and apparently the same stage has been nearly reached by the iron-mining industry of the country as a whole, notwithstanding the rapid development, here and there, of virgin fields.

Discussion.

President John Birkinbine, Philadelphia, Pa.: This paper brings up a feature which has more than once attracted the attention of those familiar with our iron-ore deposits—the peculiar ability which some of our grand-fathers and grand-uncles seem to have had as ore-discoverers. The older deposits through Pennsylvania and some of the older ones in New York State are instances; and it seems as if these men, without the geological researches and without the chemical knowledge that we have access to, had somehow picked out the best deposits. The history of such deposits as Cornwall, in Pennsylvania, some of the mines of the Lake Champlain district, and also in what is known as the Salisbury region in Connecticut and eastern New York, indicates that most of the older mines are the best mines; in other words, that in many instances the best mines are often the oldest.

In connection with Prof. Smock's paper, I would call attention to the circumstance that of the 14,500,000 tons of iron-ore produced in the United States in 1889, seventeen and a half per cent. was magnetite, and about the same proportion was brown hematite; the greater proportion was red hematite, and less than three per cent. was carbonate. Of the magnetite produced in this country, New York leads with thirty-seven per cent.; Pennsylvania comes next with a fraction over thirty-four per cent.; New Jersey is third with sixteen and one-half per cent.; Michigan fourth with ten per cent. Another point worthy of discussion is that the possibilities in magnetic separation of iron-ores may bring into a large producing capacity some of the leaner deposits which have been practically abandoned. I lately visited one of these New Jersey mines, where a narrow vein is worked at heavy expense, owing to the fact that but a small width of available shipping ore was found. The owners have lately introduced a system of magnetic concentration, by which they claim that they are so far able to reduce the cost of the ore on cars, by adding to their available supply this concentrated ore, as to transfer the mine from an unsatisfactory speculation into a profitable business.

E. C. Pechin, Roanoke, Va.: A serious trouble in concentrating a good many of the magnetic ores is the exceedingly fine condition in which the resulting product would be left. Take, for instance, the Cranberry mine in North Carolina. There is a very large deposit, that has practically amounted to nothing in the past. Beyond question, if that ore were crushed it could be concentrated to be rich in iron. But much of the concentration would be a powder, nothing like

as coarse as the concentrates of the Lake Champlain region. What are you going to do with it? Now this is the question I want to ask: Is there any objection to putting that fine stuff in the form of briquettes, assuming that it can be done commercially? Would the fact that it had been compressed into a hard block make it more difficult to reduce in the furnace than if it were simply in the shape of lumps? I have talked to a good many people about this, and they could not give me any positive answer. I should like some of our members to give their views—first, as to whether there is any objection to putting fine rich ore into briquettes by reason of its becoming less reducible in the furnace; and, secondly, as to what, in their opinion, would be the cost of so doing. Would it render the product unmarketable?

THE PRESIDENT: It has been the intention of the council to call a discussion upon magnetic separation in the future, but, in connection with Prof. Smock's paper upon New Jersey, where the feature of magnetic separation will have so much to do with the future of iron-ore mining, this may be a good time to have any of our members answer Mr. Pechin's inquiry. As preliminary to such answer, I will merely state that at Edison's works, at Ogden, New Jersey, they are now crushing their ore to 60-mesh. I doubt if any of us will claim that 60-mesh is as satisfactory for a blast-furnace as 6-mesh. It is reported on excellent authority that a sale of 100,000 tons of that 60-mesh concentrate has been made to one of our large steel works, for blast-furnace use.

R. W. RAYMOND, New York City: The paper by Mr. Nau, on experiments made at the Croton magnetic mines with the Imperatori process, indicates a use for these fine rich ores in the form of briquettes, and gives an estimate of their cost. I need not give particulars. The paper is one of those presented at this meeting, and printed copies are here.

A very interesting point is suggested by the remarks of Prof. Smock and the comment of our President as to the circumstance that in New Jersey, New York and Pennsylvania the best mines are the old ones begun by our fathers. It seems to me that in this matter, as in a good many others, we erroneously exaggerate the skill of our predecessors. Thus we hear what wonderful steel the Japanese and the Tartars could make, and how we have not got so as to make steel any better, with all our science; and now we are led to infer, from the figures given by Prof. Smock, that our forefathers had a wonderful knack of finding the best mines. The solution seems to me very sim-

ple. It is a pure case of "the survival of the fittest." You cannot estimate the amount of skill involved in any operation of metallurgy or mining by the ancients until you take into consideration the amount of waste, failures and rejections. Take, for instance, Japanese steel, which is still made after the ancient method. The ingots produced vary indefinitely; so that nine may be of poor quality, while the tenth is suitable to make a wonderful sword. The remarkable result you see, is produced by great variation first, and selection afterwards. This is the natural method, but not the scientific one. Science means to know what you want to do, and to go ahead and do it, and not have a scrap-heap as the larger part of your product. Now I am very familiar with many mines in New Jersey, at Ringwood particularly, and about Chester, where Messrs. Cooper and Hewitt own representatives of the various classes to which Prof. Smock has referred. On the Ringwood estates there are mines producing high-grade ore in large bodies, and dating back more than a century. But, on the other hand, we find upon that tract of 20,000 acres no mine our forefathers did not dig into, and we see that these big mines which, still surviving, attract attention to-day, are simply a few out of the innumerable mines which they found, not by wit, but by blind industry. We do not discover any new attractions with the needle that were not already found by the old Ringwood Company away back before the Revolution. Sometimes we strike into underground workings of theirs of which we had no record. They prospected everything, and hence the mines that are still good are old, of course, because all the mines, good and bad, active or abandoned, are old. Our forefathers had the ordinary common sense to prove all things and hold fast to those that were good. But I do not think they showed any mysterious ability, peculiar to them, in the discovery of good mines. It seems to me that if the members of this Institute were turned into an entirely new country they would probably have neither more nor less good luck than our fathers had. They would dig into everything, and the mines that turned out well would be prosecuted, and those that did not would be abandoned; and somebody might say in future, "What wonderful fellows those were that found nearly all the good mines!"

A. B. Wood, Detroit, Mich.: Most mines that turn out to be valuable mines declare their intentions very early in their development. It is seldom necessary to go through a very long siege of prospecting to find what makes a good mine. Subsequent adventures in mining districts are very often started on the basis of inferior promises, and

therefore fail, while the earlier enterprises keep on and continue to be profitable.

MR. BIRKINBINE: I did not claim any peculiar or mysterious mental characteristics for our grandfathers, but drew upon a knowledge of the history of mines throughout the country, and of instances where excellent ores, of a quality which would be called particularly good now, have been constantly exploited since their discovery many years ago. When compared with ore from other mines which have been opened since, we find the selection of the earliest operators seems to have been made with a great deal of skill, both as to quantity and quality. Among the Lake Superior mines, the two which claim to be the oldest are still among the largest and best producers, and some of the latest and best new discoveries have portions of the older operations.

THE PHYSICAL AND CHEMICAL EQUATIONS OF THE OPEN-HEARTH PROCESS.

Discussion at the Cleveland Meeting, June, 1891, of the paper presented by H. H. Campbell, at the New York Meeting, September, 1890.

HENRY D. HIBBARD, Pittsburgh, Pa.: This is a most interesting and instructive paper, some of whose lessons should find immediate application. It points out how fuel-consumption is to be decreased in open-hearth practice, by changes in the plan of the furnace.

To look first at the greatest item of loss, radiation, it is seen that thicker brick-work is advisable to lessen this. Especially is this true of the roof, made at present, almost universally, only 9 inches thick to prevent cutting. Now that high roofs, well above the course of the flame, are becoming general, and the average melter is better able to judge the temperature of his furnace, that reason has less force; and we may build roofs 18 inches thick, or perhaps better cover the usual 9 inches with an equivalent layer of sand, which has the advantage of following the arch as it rises and falls with the different degrees of heat.

The side-walls of the hearth, usually 13½ inches thick, may also be thickened up to 18 inches, and the door-linings to 7 inches; and the heat thereby saved will be valuable, while its absence from the working-platform will be much appreciated by the workmen.

To reduce the temperature of the outgoing gases, so as to cause more of their heat to be returned to the furnace, the regenerators must be made larger. This will also lengthen the life of the cast-iron of the reversing-valves and dampers.

With this increase of regenerative capacity and lower temperature of waste gases in the chimney, the height of the latter must be increased to give proper draft, while its diameter may be decreased, as the cooler gases have less bulk.

The furnaces at Steelton, from which Mr. Campbell draws most of his data, are unique in their construction and operation. They are quick melters, and work with unusually small waste, as is shown by the small amount of FeO in the slags, just after melting.

The reduction of metallic iron from the ore is also unusual in charges having so low a proportion of pig as was used in the series of nineteen heats given in Section 53, and is probably due to the same causes which give low percentage of waste; that is, unusually low oxidizing conditions. The rather long time, two and a half hours, taken in burning out half of one per cent. of carbon, and the presence of several points of manganese before tapping the steel, testify to the same thing.

In most works where pig forms not more than one-quarter of the charge, the ore added serves merely as a source and carrier of oxygen to the bath, and not as a source of iron, none being reduced to the metallic state. It is a source of oxygen to the extent of giving up that made available in reducing the higher to the lower oxides. After this initial amount of available oxygen has been used, the action is continued by the continual formation of the higher oxides Fe_3O_4 or Fe_2O_3 at the upper surface of the slag and their reduction to relatively lower oxides, FeO or Fe_3O_4 at the lower surface, where they come in contact with the metal as the mechanical action of boiling, so called, keeps the slag in motion. Without boiling, oxidation is very slow.

The results given in Section 60 c, can only be explained, it would seem, by assuming that the samples of the products of combustion at the top of the gas-checkers did not represent the average of the gases. The results obtained by analysis of the products after passing the checkers where the gases have been considerably mixed are more likely to be reliable, and certainly are more in accord with what one would look for.

It is usually expected that combustion will continue down into the checkers; but to keep this up, there must be oxygen there to do

it. It is not there in the form of steam, for no increase of hydrogen is seen in the gases after passing the checkers. Carbon is apparently taken up from that deposited in the regenerators.

It seems impossible to avoid the conclusion that the products of combustion at the top of the checkers must really have had much more oxygen than is given, and that therefore the samples were not representative. The theory that leakage of the chamber-walls supplies it is untenable.

A flame is necessarily heterogeneous in composition, but the course of the flame through the furnace is quite uniform. Now if all the samples were taken from one place it might happen that, at that particular place, by reason of local causes, the gas and air had mixed with some excess of gas and the air was practically wholly consumed, while elsewhere oxygen existed in large excess.

If there is anything in this idea, it could be found out by repeating the work with samples taken from different places at the top of the regenerators.

Referring to Section 80, the FeO taken from the bottom should be, with regular running, offset by the same amount left in the furnace, and may therefore be disregarded. This, if so, will reduce the apparent reduction of FeO during the period A B. Until a bottom is old and thoroughly impregnated with slag, there is an absorption of slag, and consequently of FeO, for every heat.

MR. CAMPBELL: In reply to the remarks of Mr. Hibbard, I would endorse his suggestions concerning thicker roof and walls. It will be urged by many persons of practical experience that this change will lessen the life of the brickwork, owing to the absence of cooling conditions. This argument is of less value than is generally believed. The wear of brickwork on the inside of a furnace may be due to any or all of at least four causes.

1. An excessive temperature throughout the melting-chamber, due to carelessness or ignorance of the melter.

2. The impact of a wandering tongue or point of flame upon some particular area.

3. The scorification of the surface due to the fine spray of metallic iron which is spurted upward from each bursting bubble of the boiling bath. These minute drops burn almost immediately and the mist of iron oxide produced is carried along by the moving gases. Part of it goes down the portes without coming into contact with any surface of brickwork until the temperature of the environment is so low that no chemical interaction follows, and the only effect is the

clogging of the passages in the checkers. Part, however, is wafted by the gaseous currents against brickwork at such a high temperature that chemical union occurs with formation of a more or less liquid slag.

4. The scorification due to the globules of ore, slag, and partially-oxidized metal resulting from the splash in adding lumps of ore and the small explosions accompanying their incorporation into the slag.

Against none of these agents is a thin wall more than a weak protection. The heat-conducting power of brick is so slight that a silica brick of the best quality will melt or scorify to a thickness of one-eighth of an inch when its outer surface is cooled by a free circulation of air or by contact with water-cooled metal. If such action can take place, it seems almost absurd to discuss the relative conductivity of nine and thirteen inches. The instances familiar to every practical man where an old, thin roof has resisted wear and tear and which have been kindly credited to radiation and conduction, will, upon careful consideration, be found due to a reduced intensity of working in the last of the run, or to more careful watching of known weakness. In any event, they can be paralleled by other cases where a new roof has lasted an equal number of charges and been found "as good as new." It will happen, however, in the case of a large area of thin overhanging roof, that the covering of a spot with brick or sand will result in the collapse of the area thus treated. But it will often be found that this comes not from increased wear beneath the covering, but to the heating and softening of the bricks through the outer surface, the consequent weakening of the arch at the spot by the absence of the cool and strong outer skin that existed before, and the concentration upon the weak spot of the movements arising from expansion and contraction. This would be the danger in using a sand covering. The objection does not apply to a roof of increased thickness properly built of brick.

If the furnace is overheated, the roof will "string" impartially in the place where it is two inches and where it is nine inches. If the point of flame impinges steadily against a spot, it will matter little whether that place is one inch or twenty in thickness. Under the conviction of the above reasoning, the writer more than two years ago designed and operated a furnace with walls and roof from thirteen to thirty-six inches in thickness, and experience has confirmed the correctness of the argument.

Mr. Hibbard mentions the indirect oxidation of the carbon and

other elements through the successive formation and reduction of different oxides of iron. The author is also in receipt of a private communication from Mr. A. D. Elbers, regarding this action, advancing arguments which may be briefly summarized as follows: Since oxidation is carried on by Fe_xO_x, the calculations for internal combustion should be modified, as the action is mainly superficial and the heat produced is carried away by the flame. It will also follow that the calorific power produced is less than the original calculation by the amount necessary to dissociate the iron oxide.

In regard to the above issues, the author would say that he was neither ignorant nor heedless of the fact of oxidation through the mediation of iron oxide. The seeming neglect to dilate upon the point was due to two reasons:

First. The difficulty if not impossibility of obtaining definite and quantitative data.

Second. The belief that these transitory conditions do not vitally affect the general thermal equation.

It should however be acknowledged that the criticism of Mr. Elbers is very just, albeit it arises from a careless use of words by the author, rather than a scientific error. By the phrase "heat from internal combustion" was meant not simply the combustion in the mass itself, for no one would suppose that carbon would oxidize to carbonic acid in the interior of the bath. The condition which it was intended to designate was the heat produced by the combustion of the elements of the bath. Whether these are burned directly and whether they are burned in the metal or as gases in the combustion chambers, cannot alter the sum total of the power which they must produce and which will exhibit itself somewhere.

From the time the gas enters the flue of the chamber until it goes into the stack—from the time the metal is charged in the doors until it flows from the tap-hole—there is a constant atomic interchange between the elements involved. The journey of oxygen through iron to carbon, is but one of them. The very oxygen so transferred was probably taken by the iron from carbonic acid, since little free oxygen should be present on the surface of the bath. A new calculation would therefore be indicated for the calorific value of such combustion. The complicated equations cannot be written which would record the chemical and thermal histories of the lower and higher oxides of carbon, manganese and iron. All this is unnecessary. Heat given to the gases is taken up in the regenerators and sent back on the next reversal, or appears in the products of com-

bustion in the stack. The reaction may alter the position in time or space of certain minor factors, but they cannot alter the sum total of heat produced from a given amount of actual and potential power without a violation of the law of the conservation of energy.

Mr. Hibbard's remarks concerning the absorption of FeO by the bottom during the first few charges and the subsequent absence of any appreciable secretion or contribution, are perfectly true of ordinary acid open-hearth work; but a reference to sections 69 and 70 will show that the estimates of amounts of FeO taken from the bottom are not founded on a neglect of the above general facts but on a quantitative estimate of a special history of deposition and erosion of infusible slag in basic practice.

A point which should have been considered in the original paper is the loss of iron in the shape of metallic globules in the slag. Determinations of the amount of shot in various samples of basic slag gave the following results:

1st sample, 3.3 per cent. 2d sample, 5.0 per cent. 3d sample, 6.5 per cent. 4th sample, 9.9 per cent.

The first sample was from a thin slag; the second and third from slags of medium viscosity; the fourth was the deposit formed on the upper part of the ladle by the spatterings of an average charge. It will be seen from section 72 that the amount of slag obtained was as follows:

	Metal.	Slag.	Per cent.
First series, ...	73,600	9,350	12.7
Second series, ...	93,620	13,140	14.0
Third series, ...	197,720	30,780	15.6

Since the average slag contains from 5.0 to 6.5 per cent. of shot, and since the weight of slag is about 15 per cent. of the charge, it is evident that in this basic work there was thus carried away in metallic globules from three-quarters of one per cent. to one per cent. of the total charge.

In acid work the loss is not as heavy. Two determinations of ordinary acid slag gave shot content as two hundredths of one per cent. of the weight of slag. The percentage of metal lost would be only about one twelfth of this, or an amount too small to be worthy of notice.

ALUMINUM IN STEEL INGOTS.

BY JOHN W. LANGLEY, PITTSBURGH, PA.

(Cleveland Meeting, June, 1891.)

THE papers of Mr. W. J. Keep, read before this Institute, have called attention to the influence of aluminum in cast-iron and on iron and steel castings. The information in these papers is interesting and valuable, as showing the modifications which iron undergoes when dosed with sufficient aluminum to form a distinct alloy, the quantity of the alloying metal used being such that the amount remaining in the casting is sufficient to be readily discovered by analysis. The same is also true of the so-called aluminum-steel patented by R. A. Hadfield, where the proportions specified are such as to leave in the finished steel from one-twentieth to one-tenth of one per cent. of aluminum.

Whether these alloys are of practical utility or not has thus far not been proved by any extended tests on a large scale, and until such evidence is forthcoming it may be safely held, provisionally, that such a use of aluminum is of doubtful utility. But there is another function which this metal can perform that has been known for some time, and which has passed beyond the experimental stage, viz.: its beneficial effect in securing sound ingots.

It appears to be a general property of all metals when in the fluid state and at a temperature considerably above their points of fusion, to contain gas, either dissolved from the air or from the furnace, or else self-evolved by internal reactions. Silver, perhaps, shows this power in the highest degree, and mercury and lead in the least; midway stands steel—the quantity of gas which it can hold in solution, or "occlusion" as it has been called, varies with its temperature, its composition, and its mode of manufacture. When the steel sets in the mould, a portion of this gas escapes, causing blow-holes, surface defects, boiling-over, wildness, and quite a long train of troubles, familiar to all practical steel-makers.

Ferro-silicon has long been used as a quieting agent. The first extended use of it for this purpose was made at Terre-Noire, in

France, some 10 years ago; now it has been discovered that aluminum will bring about the same general result, only far more effectively, and without leaving any appreciable portion of itself in the finished steel.

The writer does not know to whom the honor of this discovery is due. Aluminum has been used in this country and in England to quiet steel, so that to-day it is being used regularly for this purpose by the few steel-makers who have learned to rely on its beneficial influence. Its employment is extending, and will certainly grow in proportion as the knowledge of its power is disseminated, for there is nothing which can compare with it in efficiency, simplicity, and certainty of action; besides, it has the great advantage over silicon, that no sensible quantity of it need be left in the metal, and if used properly no impairment of the steel ever takes place.

The practice in pouring ingots is as follows: The aluminum, in small pieces of a quarter- or half-pound weight, is thrown into the ladle during the tapping, shortly after a small quantity of steel has already entered it. The aluminum melts almost instantaneously, and diffuses with great rapidity throughout the contents of the ladle. The diffusion seems to be complete, for the writer has never seen the slightest action indicating want of homogeneity of mixture—all of the ingots poured from one ladle being precisely alike so far as the specific action of the aluminum was concerned.

The quantity of aluminum to be employed will vary slightly according to the kind of steel and the results to be attained. For open-hearth steel, containing less than 0.50 per cent. carbon, the amount will range from 5 to 10 ounces per ton of steel. For Bessemer steel the quantities should be slightly increased, viz.: 7 to 16 ounces. For steel containing over 0.50 per cent. carbon, aluminum should be used cautiously; in general between 4 and 8 ounces to the ton. If these statements are put in the form of percentages, it will at once be seen how extremely minute is the quantity of aluminum which causes such marvellous results—for the numbers are:

Four ounces equals	0.0125	per cent.	$= \frac{1}{8000}$
Five ounces "	0.0156	"	$= \frac{1}{6400}$
Eight ounces "	0.0250	"	$= \frac{1}{4000}$
Sixteen ounces "	0.0500	"	$= \frac{1}{2000}$

To explain the reason of the ranges in the limits given for the amounts of aluminum which experience has shown to be effective,

a slight discussion of the theories of its action may not be out of place.

The gas contained in melted steel seems to be held by two distinct forces, or perhaps it would be better to say by one force manifesting itself in two degrees. First, there is apparently a part of the gas held in simple solution, which begins to escape the instant the metal is tapped or mechanically agitated, either within or without the furnace. This may be compared to the carbonic acid which causes the effervescence of soda water as it escapes from the hydrant of the druggist. In this case the foaming is due to a release of pressure, while in the steel it is owing to a commencement of cooling. Second: another part of the gas is perhaps really occluded; at any rate it is not given off till the steel begins to crystallize. It is this portion which causes most of the blow-holes, and makes *wild* steel; it may be compared to the dissolved air in water, which causes the bubbles that nearly all surface-ice shows. The weight of this air cannot exceed $\frac{1}{20000}$ of that of the water, and yet it is sufficient to render ice "unsound" in the steel-maker's sense of the word.

To show how very little gas will make the difference between a good ingot and a bad one, the writer tried the following experiment: A mixture which would slowly give off gas when heated was spread in a thin layer on the interior of a mould destined to make a 90-pound steel ingot. The metal was such that in a clean mould a sound ingot, having the necessary indication of soundness, viz.: a *pipe*—that is, a conical central cavity extending from the top to about one-fourth of the length down—would have been formed. In the prepared mould, however, not only was there no pipe, but the ingot boiled and rose some two inches. On breaking it it was full of large blow-holes. Now the mixture on the interior of the mould could not have produced more than 30 cubic inches of gas if it had all been set free; this would have been only $\frac{1}{64800}$ of the weight of the ingot.

It is apparent, then, that the actual weight of the gas given off at the moment when the steel "sets" or crystallizes, is exceedingly small, yet its evil effect is very great. The aluminum appears to act on this retained, dissolved, or occluded gas, and in some way prevents it from being liberated. Granting that the aluminum does really operate on this portion of the gas, we need not be surprised at the small quantity of the metal necessary to be used, because the agent on which it acts is present in a still smaller proportion. The practical effect, however, is none the less wonderful when we see an

ingot which would have boiled over to the extent of one-fourth of its volume in the absence of a stopper, perfectly quieted so that it lies dead in the mould without stopper or sand, from the effect of one half-pound of aluminum.

The very thoroughness with which the work is done constitutes a danger to be guarded against in the use of aluminum, that of causing excessive piping. Most crystalline bodies when they pass from the fluid to the solid state, provided they are free from contained gas, form pipes when cast into moulds; the amount of the pipe is dependent on the nature of the substance, on the way it is poured, and on the form of the mould. This is easily seen in zinc, steel, some kinds of brass, indeed most metals and alloys, also in non-metallic substances, as stearic acid, paraffine, sulphur, etc. If there is also gas present the disposition to pipe is lessened with the concomitant tendency to form blow-holes.

It seems to be inevitable in steel practice that either a pipe or blow-hole should be formed. Now it would appear that the best results are attained when the joint effect of both pipe and holes is reduced to a minimum; this is secured by using just enough of the quieting agent to allow the top of the ingot to rise a little and, therefore, to leave in it only a small pipe. By this procedure the slight inevitable unsoundness will be generally confined to the top of the ingot.

Since the quantity of aluminum to be used varies with the initial wildness of the steel, as well as on its temperature and on the size of the mould, it is apparent that each steel-maker must determine the best proportions for himself, but it is believed the limits previously given of from 4 to 16 ounces of aluminum to the ton of steel will cover all cases likely to arise in practice.

Two theories have been advanced to account for this action of aluminum; one of them is that the metal acts by taking oxygen away from the carbon oxides dissolved in the steel, thus changing the gas into two solid products, carbon and aluminum oxide. The other view holds that aluminum and silicon each act by increasing the solubility of the gases in the steel, so that none will be given off at the moment of setting.

The arguments for these views are briefly these: When only a small quantity of aluminum is used, say four ounces, it is difficult or impossible to find it in the resulting steel. If larger amounts are taken, the portion remaining in the steel is always small, and in this case a faint whitish blush of aluminum oxide can often be seen

in the interior of the pipe, or even on the surface of the ingot, thus proving some oxidation of the metal. On the other hand, the adherents of the solution theory point to the undoubted fact of hydrogen being found occluded in steel, also nitrogen and perhaps other non-oxidized gases; these they say can neither combine with aluminum to form hydrides and nitrides, nor can the metal decompose them since they are elementary bodies.

May it not be the case that both of these views are partly correct? Two facts would seem to indicate it, because when the minimum of 4 ounces is used no external coating is seen on the ingots; on the other hand, when enough aluminum is taken to produce the coating the ingots are always piped very deep, and they are more liable to defects from surface pits and scales, owing to the infusible film of aluminum oxide. It may be taken as a safe rule that when the alumina coating can be seen on the surface of an ingot too much of the metal has been taken and the limit of benefit has been overpassed.

One unexpected action of aluminum has quite recently been discovered by Mr. George G. McMurtrie, President of the Apollo Iron and Steel Company, viz., that it could be made to replace manganese. This seems to me to be a wholly new fact in the metallurgy of steel, for this gentleman has rolled ingots down to thin sheets by using one half-pound of aluminum. The sheets, samples of which are before you, show either no manganese or only traces; none was added in the process of manufacture.

I quote from a recent letter received from Mr. McMurtrie:

"I take great pleasure in stating that we were quite successful in making a heat of open-hearth steel without the use of manganese, and send you by express two bars and two sheets which you will notice were successfully rolled.

"Having watched the operation clear through from the charging of the furnace to the casting of the ingots, which were subsequently hammered from 12 x 12 inches down to 6 x 6 inch billets, then rolled into sheet bars and then into sheets, I can say without hesitation that the entire operation was a practical success. Our ingots were as solid as any we have ever made. The molten steel lay as quiet in the mould as water.

"In hammering we took a strong heat, having the billet finished before the red short danger line (cherry-red) was reached. The sheet bar was rolled without the slightest difficulty or excessive waste, and the only contingency in rolling the sheets was the extra

loss of 4 inches in width, which was caused by the steel cracking to that extent more than is usual with our ordinary material.

"The amount of aluminum used in the ladling was a fraction over a half-pound per ton of steel. . . . . We have sent samples of this material to Prof. Thompson, Electrical Engineer of the Cornell University, to test for electrical purposes."

In a subsequent letter Mr. McMurtrie says: "We rolled a lot of No. 30 gauge sheets a few days ago from steel made by substituting aluminum for manganese, and must say that the rolling was done in a most satisfactory manner, showing that for any purpose when manganese is an objection its use may be avoided by substituting one-half pound of aluminum per ton of steel. . . . . I might also mention that one-half pound of aluminum will so affect 20 tons of molten zinc as to make a marked difference in the appearance of galvanized sheet-iron, and continues to show its effect for probably 2 to 3 hours while constantly dipping the sheets. As soon as the effect becomes unnoticeable 3 ounces will restore the peculiar lustre.

What little we have done has been prompted by an instinctive feeling that it cannot injure the quality of any other metal when used carefully, while the chances are in favor of making a decided improvement from the fact that it will make any metal more fluid and castings more solid."

These observations of Mr. McMurtrie show a novel use for aluminum.

In conclusion, the writer believes that the use of aluminum in steel ingots for quieting purposes is not covered by, and does not infringe the Hadfield patent.

The claims of Hadfield for his aluminum-steel require that one-twentieth to one-tenth of 1 per cent. should remain in the steel as an alloy. This would be 16 to 32 ounces to the ton as a residual quantity, while it will be seen the amounts used for the purpose of quieting steel are less than the minimum limit claimed in the Hadfield patent.

Discussion.

R. W. Raymond, New York City: I would call attention to the very interesting discussion of Mr. Hadfield's paper on "Aluminum-Steel," which will be reprinted, with the paper itself, in vol. xix. of the *Transactions*. Among the contributions to that discussion is one by Prof. J. O. Arnold, of Sheffield, England, parts of which may be considered as a spontaneous and independent indorsement of Prof. Langley's conclusions (*Trans.*, xix., 1088).

PROF. LANGLEY: I have seen the paper, but not the discussion.

HENRY D. HIBBARD, Pittsburgh, Pa.: I would ask Prof. Langley what he considers becomes of the additional shrinkage that he shows in the top of his ingot.

C. E. STAFFORD, Pittsburgh, Pa.: I would also like to ask Prof. Langley as to the amount of sulphur and phosphorus, since that would have considerable to do with the behavior in rolling; and also as to his whole ingots, whether he has broken them, and, if there are any blow-holes in them, in what sections they appear; also whether he has analyzed different portions of the ingots, and whether there is any separation of the sulphur and phosphorus, so as to make a difference between the outside and the inside?

PROF. LANGLEY: The greater part of the blow-holes will be found above the horizontal plane passing through the middle of the pipe. I have broken quite a number of ingots, and while most of the blow-holes are there, I am bound to say that occasionally you do find them down below. As to what becomes of the additional shrinkage, that is a difficult question. I will answer it, therefore, by a provisional hypothesis. When enough aluminum is put in, so that there is no rise of the top of the ingot, the cavity of the pipe represents the shrinkage, due probably to the difference of temperature between the fluid metal and the metal after solidification. When enough gas is allowed to stay in the ingots so as to cause incipient boiling, there is always a rise here (indicating on blackboard); also a portion lying between the apex of the pipe and the top of the ingot. Of course, there is the same total amount of shrinkage there in both cases.

Mr. Stafford's other question refers, I suppose, to those plates of Mr. McMurtrie's. Those were sent to me only a short time ago. I have only made the analysis, the figures of which I have given. I understand from him that it is the regular stock that he is normally producing in the ordinary use of manganese, only that in this case no addition of manganese was made, but in place of it the half-pound of aluminum per ton was put in. I have not made an analysis of this material, nor have I made an analysis of the ingots to ascertain whether there was any more segregation of sulphur and phosphorus in an aluminous than in an non-aluminous ingot.

JOHN G. SPEER, Pittsburgh, Pa.: Can Prof. Langley say whether aluminum has been tried in boiler-plate steel, and how it affects the tensile strength and the reduction of area? It seems to me that the

advantage of having solid ingots for boiler-plates is very great indeed; and while we accept the candid admission that aluminum does not remedy surface-defects, at the same time the getting a thoroughly solid ingot would be a very great point with us.

PROF. LANGLEY: The steel which I have been interested in making has all been destined ultimately for machinery or for tools. Therefore, I do not know how it would work in boiler-plate. In the majority of cases the carbon was over fifty-hundredths.

R. C. COLE, Cleveland, O.: I would like to have Prof. Langley say whether this work that he did and that of Mr. McMurtrie was done in pure aluminum or with a mixture of pure aluminum and ferro-aluminum? Both metals are sold on the market, and both are used for this purpose.

PROF. LANGLEY: The work in both cases was done with so-called commercially pure aluminum, containing upwards of 95 per cent. of the metal.

MR. COLE: In a conversation which I had with Mr. Morris in Pittsburgh, about five months ago, he told me that he believed he could use aluminum in the place of manganese to a large extent; and two or three weeks ago he told me that the results of his work up to that time were very satisfactory, but that he was not prepared then to give any data about it. Prof. Langley says he does not know to whom the honor is due of making this discovery of the use of aluminum in steel. *Apropos* of this meeting in Cleveland, I would like to say that I think the first work that was ever done in the matter of trying to remove blow-holes from steel by means of aluminum was done here. Whether it had been done before abroad or not, I am not aware, but it was done at the Cleveland Rolling Mills. I had the pleasure of being present at the experiment. We made some castings in sand from an open-hearth furnace of steel to be rolled into plates, and we added to one cast about one-twentieth of one per cent. of aluminum before pouring into the mould, while we took steel from the same bath, without adding the aluminum, and poured it into other moulds, and we distinctly saw a great difference. The blow-holes in the ordinary steel all seemed to be radial; the samples were cylindrical, and the blow-holes seemed to be radial and at the periphery. And in the case of the aluminum, it was perfectly solid in every case. There was not a blow-hole to be seen anywhere, while in every case where the aluminum was not used the blow-holes were very marked. That was done in the early part of 1886, at the Cleveland Rolling Mills.

GUILLIAEM AERTSEN, Latrobe, Pa. (communication to the Secretary): I think the experimental furnace of Mr. Ostberg, at Worcester, Mass., was in operation before 1886. The Cleveland experiment seems to have been antedated also by Mr. R. W. Davenport, then Superintendent of the Midvale Steel Works, Philadelphia. Mr. Davenport made, December 26, 1885, a number of ingots of crucible-steel to which aluminum had been added, and July 27, 1886, he conducted a more elaborate experiment in the open-hearth, with very marked and interesting results. See Howe's "Metallurgy of Steel," p. 87, foot-note.

W. J. KEEP, Detroit, Mich.: I have said, in my paper already published in the *Transactions*, all that I have to say on the effect of aluminum in iron and steel. I may here mention, in addition, a little experiment, which shows how directly aluminum mixes with iron. I took a ladle of cast-iron and threw on top of the molten metal a piece of aluminum (one of the first pieces of pure aluminum I had) that was about the size of a pea. Instantly it permeated the whole mass and set the metal boiling, of course liberating the impurities; but unfortunately the impurities were carried down into the metal again by the violent boiling. This metal was poured out into the mould, and 50 pounds more were caught in the same kettle, and the action of the aluminum remaining was exactly the same, so far as I could detect, as in the first ladle. After that another 50 pounds was caught, and the action was less marked, but it still appeared, and even a slight film was apparent upon the surface of the metal. It seems to me this ought to settle the question whether aluminum will form a perfect union with iron in any shape. The same thing was apparent in an experiment at the Michigan Stove Co.'s Works, where we took a crucible holding 80 pounds of molten raw iron and put in successive additions of aluminum. Each time it immediately went through the whole mass; and the metal, even after the sixth addition—although the crucible stood on a cold iron platform—seemed to be much more fluid than it was when first poured.

INTERNATIONAL STANDARDS FOR THE ANALYSIS OF IRON AND STEEL.

[In continuation of the Paper presented at the Pittsburgh International Session, October, 1890, and printed in *Transactions*, xix., 614.]

BY JOHN W. LANGLEY, PITTSBURGH, PA.

(Cleveland Meeting, June, 1891.)

IN the report of the American Committee, made last October, reasons were given for calling in question the accuracy of all carbon-determinations in iron or steel, made by the customary mode of dissolving the metal in double chloride of copper and ammonium; but the Committee could not at that time feel sure that the source of the error had been detected.

The members of the Committee engaged in this investigation are now able to say that all of the double chlorides of copper and ammonium which they have examined contain tarry matter (probably pyridine compounds), held in solution in all commercial ammonia. This substance is retained by the carbon-sponge, and hence causes the apparent carbon in the steel to be reported too high.

A complete remedy for this trouble is found to be the substitution of chloride of potassium for chloride of ammonium, and the making of the solution slightly acid.

They recommend dissolving the metal in a solution of double chloride of copper and potassium, to which 5 per cent. of its volume of strong hydrochloric acid has been added. A neutral solution of the double chloride gives too low results, and is objectionable.

THE DIRECT DETERMINATION OF ALUMINUM IN IRON AND STEEL.

BY THOMAS M. DROWN AND ALEX. G. MCKENNA, MASS. INSTITUTE OF TECHNOLOGY, BOSTON, MASS.

(Cleveland Meeting, June, 1891.)

THE unsatisfactory character of most, if not all, of the processes for the direct determination of alumina in the presence of iron and phosphoric acid, and the sharpness with which both the iron and phosphoric acid can be determined, have led chemists to rely mainly

on processses in which the alumina is obtained by difference. But in the modern alloys of aluminum and iron, where the aluminum may be present only to the extent of a small fraction of one per cent., nothing short of the isolation of the alumina itself can give satisfactory evidence of its presence.

In the course of some experiments on the electrolytic deposition of metals, we have found that it is possible to remove iron completely from its acid solution in large quantity and in short time, by a current that will not in the least affect the alumina in solution. The essential condition of success in this operation is the use of mercury as a cathode, the iron forming an amalgam with the mercury as fast as it is deposited from solution. This use of mercury was proposed by Dr. Wolcott Gibbs, in a paper read before the National Academy of Sciences in 1883, as a method of determining many metals by electrolysis which are not at all, or only imperfectly, deposited on a platinum cathode. It is certainly one of the most valuable and suggestive contributions to the quantitative separation and determination of metals by electrolysis that has yet been made.

The following are some of the experiments we have made to determine the conditions of success in the application of the method to the determination of aluminum in the presence of iron.

First, as to the completeness of the precipitation of the iron. The first attempts to determine the iron which had been removed from the solution and alloyed with the mercury showed an apparent loss, notwithstanding the fact that the iron was completely removed from solution. This was subsequently found to be due to the volatilization of some of the mercury on drying before weighing. The amount of mercury which may be thus lost is very considerable, as the following experiments show.

About 75 grammes of mercury were put into a beaker, which was then weighed. It was washed several times with water and once or twice with alcohol. The loss on drying different times and at different temperatures was as follows:

Time of Drying.	Temperature.	Loss, Gramme.
10 minutes.	140° C.	.0096
"	100°	.0043
"	"	.0028
"	"	.0030
"	"	.0030
"	"	.0025
2 hours.	70°	.0040

Time of Drying.	Temperature.	Loss, Gramme.
5 hours.	70°	.0120
20 "	"	.0520
48 "	25°	.0030
48 "	24°	.0025
24 "	"	.0007
24 "	"	.0012
48 "	"	.0042
48 "	"	.0033

But it is, in fact, only necessary to heat the mercury for about two minutes at a temperature of 100° C. to free it completely from moisture, and in this time the loss is very small. The following are the amounts lost in nine experiments under these conditions: .0002, .0005, .0005, .0003, .0008, .0005, .0002, 0, 0, an average of 0.00033 gramme.

There seems also, at times, to be a loss not connected with the volatilization of the mercury, which, possibly, may be due to impurities in the mercury, metals, for instance, which are dissolved out by the acid, and not precipitated on the mercury again. On this account it is desirable, in cases where the process is used to determine iron or other metal, to have a blank beaker in the circuit, containing only the mercury and dilute sulphuric acid, and to add any loss which may be found in this beaker to the amount found in the others in the series.

The following are a series of results obtained by electrolyzing a solution of ferrous ammonium sulphate, slightly acidulated with sulphuric acid.

Iron Taken. Gramme.	Iron Found. Gramme.	Loss of Weight in Blank. Gramme.	Iron Found + Loss in Blank. Gramme
.2260	.2254	Not made.	
.3727	.3725	"	
.3080	.3089	"	
.3238	.3232	"	
.0647	.0632	.0020	.0652
.0647	.0634	Not made.	
.0647	.0649	0000	.0649
.0647	.0635	Not made.	
.0604	.0602	.0003	.0605
.0604	.0608	.0003	.0611
.3020	.3008	.0003	.3011
.3020	.3000	.0003	.3003

The best conditions for the rapid precipitation of the iron are a strong current (about 2 amperes, or 20 c.c. of electrolytic gas a min-

ute), a nearly neutral solution of small bulk, a large amount of mercury (not less than fifty times the weight of the iron to be precipitated), and a large anode of platinum. The mercury cathode is brought into the circuit by means of a platinum wire enclosed and fused into one end of a glass tube, which passes through the liquid. It is of advantage to pour mercury into this tube to the depth of an inch, in order to weight it and make the connection with the mercury cathode more stable.

By observing the above conditions it was found practicable to precipitate 10 grammes of iron in from 10 to 15 hours. The iron amalgam may contain as high as 10 per cent. of iron. On standing a long time exposed to the air, the iron separates out as a black powder, which may be removed from the mercury by agitation with water. Mercury which has been used for iron precipitations may be purified by removing the greater part of the iron amalgam by filtration through chamois skin, and then passing air through the filtered portion, as recommended by J. M. Crafts.

Second, as to the behavior of manganese in solution under the foregoing conditions. Inasmuch as manganese is almost always present in iron and steel, a good many experiments were made to find what becomes of it under the electrolytic conditions which we have been considering. It was found that a portion is oxidized to the dioxide, coating the platinum anode, and ultimately falling off, in great part, in scales, and that a portion is reduced to the metallic state and alloys with the mercury. This fact of the reduction of the manganese was evident on electrolyzing a solution containing only dilute sulphuric acid, using as a cathode mercury which had been previously used in an experiment with a manganese solution. In a few minutes the permanganate color appeared, and a slight coating of manganese dioxide was deposited on the anode. The experiments made to determine what proportion of manganese was reduced and what proportion oxidized were not satisfactory, but in general it may be said that about half the manganese alloys with the mercury; the other half is nearly all oxidized, and is found both on the anode and in scales floating in the liquid. A small portion is always found in solution. By using both electrodes of platinum it was found possible, in the absence of iron, to precipitate manganese completely as dioxide; but the coating on the anode was never coherent when more than mere traces of manganese were present. But the danger of the re-solution of the floating particles of the manganese dioxide, together with the necessity of filtering and igniting it to

the proto-sesquioxide, makes the process an unsatisfactory one for the determination of manganese. The best conditions were found to be a concentrated solution of manganese sulphate to which a little nitric acid had been added, a rather weak current, a small cathode and a large anode.

Third, as to the influence of phosphoric acid on the precipitation of iron. Two solutions containing equal amounts of iron were electrolyzed in series. To one was added one gramme of crystallized hydrogen di-sodium phosphate, and to the other dilute sulphuric acid. When the iron was all precipitated, the beakers with the mercury, together with the glass tubes containing the platinum wires, were weighed, and found to have gained, respectively, 0.1290 and 0.1275 gramme. To a solution of 3 grammes of iron in sulphuric acid, 0.1935 gramme crystallized hydrogen di-sodium phosphate was added, equivalent to .0167 phosphorus. After the iron was completely removed from the solution, the phosphoric acid was precipitated by magnesium mixture and .0163 gramme phosphorus obtained. A duplicate experiment gave .0161 gramme phosphorus. A third experiment with the same amount of iron gave, under like conditions, .0064 gramme phosphorus when .0068 was added. From the experiments we may conclude that the separation, by this electrolytic method, of iron and phosphoric acid in solution is satisfactory.

Fourth, as to the separation of iron and aluminum. Known amounts of iron and aluminum in solution together as sulphates were electrolyzed, and, after all the iron had been deposited, the alumina was precipitated by ammonia. Following are two results thus obtained.

Alumina Taken. Gramme.	Alumina Found. Gramme.	Iron Taken. Gramme.	Iron Found. Gramme.
.0283	.0286	.2286	.2277
.0142	.0142	.2286	.2283

The iron, determined by weighing the mercury after electrolyzing the solution, includes in both cases a blank of .0020 gramme, which was the loss of a mercury cathode in the same series, when dilute sulphuric acid only was electrolyzed.

The process was now tried under the conditions which would obtain in the determination of aluminum in iron and steel, namely, known amounts of aluminum sulphate were added to a known amount of steel. After much experimenting the following procedure was adopted as being the most uniformly satisfactory.

Dissolve 5 to 10 grammes of iron or steel in sulphuric acid, evaporate until white fumes of sulphuric anhydride begin to come off, add water, heat until all the iron is in solution, filter off the silica and carbon, and wash with water acidulated with sulphuric acid. Make the filtrate nearly neutral with ammonia, and add to the beaker in which the electrolysis is to be made, about one hundred times as much mercury as the weight of iron or steel taken. The bulk of the solution should be from 300 to 500 c.c. Connect with the battery or dynamo current in such a way that about two amperes may pass through the solution over night. This we have generally accomplished by using three lamps of 32-candle power arranged in parallel on an Edison circuit. In the morning the solution is tested for iron, and, if necessary, the electrolysis is continued after adding enough ammonia to neutralize the acid that has been set free by the deposition of the iron. The progress of the operation may be observed by the changing color of the solution. At first it becomes darker in color near the anode; after five or six hours it is nearly colorless, and finally becomes pink, from the formation of permanganate.

When the solution gives no test for iron, it is removed from the beaker with a pipette while the current is still passing. When as much has been removed as possible without breaking the current, water is added and the operation continued until the acid has been so far diluted that there is no danger of dissolving iron from the mercury. The anode is now taken out and the mercury washed with water until the last traces of the solution have been removed from it. After filtering, to remove any flakes of manganese dioxide which may be suspended in the solution, sodium phosphate is added in excess and ten grammes of sodium acetate. The solution is now made nearly neutral with ammonia and boiled for not less than forty minutes. The precipitate of aluminum phosphate is then filtered off, ignited and weighed. It should be white after ignition. If it has more than the faintest shade of color it must be dissolved by fusing with acid potassium sulphate and again electrolyzed for two or three hours. The second precipitate has been found to be always white without a trace of iron, but, in general, we have found the first precipitate to be free from iron or to contain such a minute amount that the second treatment is not necessary. We have determined that .0003 gramme of iron will give a decided red color to .0076 gramme of aluminum phosphate. The precipitate of aluminum phosphate, produced as above, does not always have the composi-

tion Al_2O_3,P_2O_5. Our analyses of it go to show that the formula $7Al_2O_3,6P_2O_5$ more nearly expresses its average composition, and in those cases in which the amount of phosphoric acid and alumina have not been actually determined in the precipitate, we have assumed this composition, which is equivalent to 24.14 per cent. of aluminum in the precipitate.

It is necessary to boil for at least forty minutes to precipitate completely the aluminum phosphate, and excessive washing of the precipitate should be avoided.

The following table gives the result obtained in determining by the above process the aluminum added in known amounts to solutions of steel.

Steel Taken. Grammes.	Per Cent. of Aluminum Added.	Per Cent. of Aluminum Found.
5	0.39	0.36
5	0.39	0.38
5	0.39	0.38
5	0.39	0.38
5	0.39	0.37
5	0.043	0.045
5	0.043	0.041
5	0.043	0.049
5	0.043	0.048
10	0.027	0.015
10	0.200	0.160
10	0.046	0.044
5	0.085	0.088

A blank experiment with the same steel, without the addition of any aluminum, gave a precipitate of aluminum phosphate equivalent to 0.004 per cent. of aluminum, which has been deducted from the results in the above determinations. This small amount came, doubtless, from the reagents and the glass vessels used, and may have been partly silica. This correction should be made in all cases where the amount of aluminum is very small.

A sample of Mitis steel analyzed by this method gave the following result:

Steel Taken. Grammes.	Aluminum Found. Per Cent.
4.4260	0.014
4.4260	0.013
12.6650	0.014

A sample of "aluminum physic" used in the preparation of Mitis castings gave as follows:

Iron Taken. Grammes.	Aluminum Found. Per Cent.
1.6130	6.28
0.4030	6.52
0.4030	6.22

A sample of pig-iron to which aluminum had been added in remelting gave as follows:

Iron Taken. Grammes.	Aluminum. Per Cent.
1.4576	0.789
1.8167	0.806
1.6390	0.794
1.6130	0.802

It might be thought that the process would be simplified by reducing the iron to the state of protoxide, and then precipitating the alumina as basic acetate, subsequently removing by electrolysis the small amount of iron precipitated with the alumina. A number of experiments proved, however, that this modification not only gave less accurate results, but involved much more work than the precipitation of all the iron by electrolysis. When the iron is all removed over night without involving any work on the part of the analyst, it is surely superfluous to go through a more or less tedious chemical operation to remove a part of the iron.

It is obvious that this process of electrolysis with a mercury cathode may be of value in other analytical operations in which the removal of the iron would be of advantage; as, for instance, in the determination of phosphoric acid by magnesia.

ELECTRICITY IN WELDING AND METAL-WORKING.

BY A. B. WOOD, DETROIT, MICH.

(Cleveland Meeting, June, 1891.)

IN welding and metal working by electricity, two systems are in use,—the so-called incandescent system, in which the material operated upon is traversed by currents of large volume and low electro-motive force, the current having a continuous metallic circuit during the welding operation, and the arc system, in which the electric arc is utilized.

It is the purpose of the present paper to call attention to this last

system of electric welding as one of the recent developments in the art of metal working.

In the arc-system of electric welding the material may be included in the electric circuit, or may be wholly without the circuit; in either instance the enormous heat of the electric arc is brought into requisition and utilized in the welding or metal-working operation.

In the Coffin arc-welding system, invented by Mr. C. L. Coffin, of Detroit, the material is in the electric circuit or independent of it, as the case may require, different processes being applied as may be best suited to the work.

It is not the intention in this paper to give a description of the various machines and appliances employed, as improvements are being perfected and the inventor prefers to hold such descriptive matter in reserve until he shall have secured the protection of patents.

The apparatus used in this arc-system of metal-working is of the simplest character. It can be used upon continuous or alternating currents, upon light- or power-circuits, either arc or incandescent.

In the earlier attempts at arc-welding with the Bernardos process, difficulty was experienced in obtaining clean and homogeneous welds. This objection does not apply to the Coffin system.

In the Bernardos process the material is connected to one terminal of the generator, and the tool, generally a carbon electrode, connected to the other terminal. Contact is made between the tool and material to establish the circuit, and an arc is sprung between the tool and material by separating them slightly after the current has been turned on. The carbon electrode is slowly traversed along the point or part to be welded; the intense heat of the arc fusing the metal at or in the neighborhood of the joint. Since the regulating and maintaining the arc at proper length is done by hand, much difficulty is experienced in securing a uniform arc, while the regulation of the dynamo is almost an impossibility; the material is subjected to too severe treatment and, in many instances, notably in working on steel or iron, a chill is formed at the weld, or the metal is overheated and greatly oxidized.

With the Coffin system of arc-welding this difficulty is entirely obviated, the material not suffering in the least from the treatment; clean, neat welds are made with but little oxidation; no chilling or burning is manifest; the arc is under perfect regulation, and the heating of the material is entirely under control of the operator. Generally no flux is used in the welding-operation, so slight is the scaling or oxidation, though, if desired, fluxes may be used in special instances, as in the welding of polished metal.

This arc-system of metal-working is of very general adaptability, being readily applied to bar, shaft, axle, ring, hoop, tire, pipe, sheet-metal, angle-iron, frame- and boiler-work. A one-horse-power arc-welder has a capacity up to one inch of round iron; its cost of maintenance is but a few cents per day for power supplied at the dynamo when power is rented, or at a rating of a single arc-lamp when current is supplied from light- or power-circuits.

A welder of one horse-power has a capacity of one weld per fifteen seconds, operating upon one-quarter-inch round iron. This capacity can be increased according to the skill and rapidity of the operator. The same welder handles half-inch and five-eighths-inch round iron with equal facility, and can be utilized upon three-fourths inch and one-inch work. The butt-welding of pipe is obtained with the same ease and perfection of results.

The arc-system of welding seems destined more especially to give material aid in work on sheet-metal, tubes, and boilers, since the tremendous heat of the arc can be readily applied where most needed. The heating of the material is not dependent upon its electrical resistance, or its current-carrying capacity, nor is perfect contact at the joint a necessity; while these are points of the utmost importance in operating upon such work under the incandescent system. With that, the current required to weld a five-foot seam in half-inch iron plates is something enormous, and the cost of generator and of power to drive it is correspondingly great; while under the arc-system the plant required would be insignificant in comparison. It is not the intention of the writer to disparage the incandescent system of electric welding, for it has great use, but to call attention to the fact that arc-welding is an established process, in successful operation, and bids fair to assume a position of great importance in metal-working operations.

The mechanical application of arc-welding is readily made. Since the material may be either within or without the circuit, and the heating-effect is independent of the current-carrying capacity of the material, no exacting provisions have to be made for electrical contact, or large current-carrying capacity in the material itself. Hence heavy copper castings and retaining-clamps are unnecessary; no cooling appliances are required, and in most instances the welder proper may be applied to existing apparatus or machinery with slight alteration. An advantage of this method may be noticed in the welding of hoops and tires, in that it does not require special precautions to prevent short-circuiting of the current around the

joint, or the use of an abnormal current to cover the effects of such a short circuit, or shunt, should it occur. A hoop of one-half inch or three-fourths inch round iron is rapidly and neatly welded upon a one horse-power arc-welder, a feat quite impossible with like power under the incandescent system.

With arc-welding of larger power the work possible to be done is much more general in its nature. A fifty-horse-power arc-welder would be an interesting piece of mechanism when in operation and the work it could handle would be of large proportions. Large shafts could be welded as readily as small bars are now treated upon small welders. It is not improbable that in the near future, rails, rail-butts, shafts, and similar large work will be welded upon arc-welders of moderate power.

The simplicity of the apparatus, its small cost of maintenance, and the wide range of work possible with a single welder of small horse-power, renders the arc-system of interest to bridge-builders, mine-managers and others, whose operations in the field do not permit extensive power-plants and generators. A ten- or twenty-horse-power arc-welder has capacity for most of the work arising in bridge-construction, for welding eye-bolts, spanner-rods, plates, etc., and for the various repairs continually arising during construction. The dynamo used for welding in the daytime can be utilized for lighting at night.

On the Secretary's table may be seen some samples of arc-welding, including inch bars, done by the writer and Mr. Coffin on a one horse-power welder. Though we were totally without experience in welding by any other means, the samples show complete success in making perfect welds. No flux was used, and each weld was completed at one heat. The bent samples are cold bent. Particular attention is called to the soft and natural condition in which the iron is left at the welds.

Discussion.

Prof. J. W. Langley, Pittsburgh, Pa.: I have not had the pleasure of seeing machines of Mr. Wood's in operation, but it seems to me that the principle here involved is certainly correct. Necessarily, by theory, these arc-machines must be more economical than those which operate by incandescence, because it is a law that the evolution of heat is in direct proportion to the resistance at a particular point. If we bring two large masses together and heat them by incandescence, at the first points of contact the resistance is

very large, but in proportion as the metal is squeezed together and the number of points of contact becomes greater, the local resistance falls off; and, therefore, just at the time when the complete contact of the parts is brought about, the local generation of heat almost vanishes, and we have only the heat which has been stored up during the earlier stages of the process. By the arc-system, on the other hand, the resistance is that of the arc, which remains constant, and the arc can be directed to all parts of the weld. As illustrating how very rapidly heat is communicated from an arc to a mass of steel, and how very little of it will be conducted away laterally into the body of the metal, I may cite an observation made incidentally about a year ago, and having no direct connection with welding. I was melting a large mass of steel in an electric-arc furnace, by placing the lumps of steel in a crucible connected with one pole of a dynamo, and lowering the positive electrode down upon the steel and swinging the arc upon it; and on several occasions it happened that the arc from the end of the carbon rod struck squarely in the center of one of these lumps of steel (about as large as a hen's egg). In a very few seconds that arc would bore a hole through that piece of steel, the corners of the steel being still so cold that, when seen through the dark spectacles that one had to wear, they appeared to be black. Thus in a few seconds the steel was raised beyond welding-heat in the center. Therefore, it appears to me that in this method of arc-welding there will be very little loss of heat, and the process seems to have, in this respect, an advantage over incandescence.

R. W. RAYMOND, New York City: The incandescent system of electric welding has been used for some time with great success at the works of the Trenton Iron Co., where we are constantly welding wire by that system. What seems to be perhaps the most delicate test of the completeness of the weld has been incidentally developed in the manufacture of telegraph-wires. As is well-known, the telegraph companies deprecate an excessive number of joints in wire, and have been accustomed to specify in their orders for wire a certain maximum number of joints to the mile, as well as to require a certain conductivity. These stringent conditions often necessitated on the part of manufacturers the purchase of rods of maximum length, at an extra price above that of shorter rods or smaller billets, otherwise of the same quality. Ordinary welding, in order to produce the required lengths of wire, was not permitted, as it increased the electrical resistance, and thus the operating cost of the line. After

we had introduced the electric weld an attempt was made to get the telegraph company to accept pieces thus welded. They said at first, as consumers always say, "Well, we are satisfied, and we don't want to be bothered with experiments on new-fangled notions," and so we abandoned the attempt. But shortly after, we sent them, included in a lot of samples of ordinary wire for test, a sample that had been cut into 52 pieces and welded electrically 52 times, instead of being a continuous piece of original metal; and that sample proved equal or superior in conductivity to all the rest. This result being verified by additional tests, there was no further objection to electric welds, and they are now actually called for in specifications. As a test of perfect metallic continuity, and also of the absence of any chemical alteration due to the welding-process, I think this is as delicate as could be asked. For the class of work mentioned, the result is perfect; and any other weld which is claimed to be as good, not to say better, should meet this electrical test to begin with.

As to the matter of economy which Prof. Langley has mentioned, he may be right on the point of theory; but the question of this particular economy is certainly not an overwhelming one, in large works where power is abundant. A little more or less steam-power is not important, compared with the saving of time and manual labor, and the securing of perfection in results.

I do not quite understand from Mr. Wood's paper in what way large masses are to be welded by the arc-method. The method is apparently one of exterior heating, which, when produced by the electric arc, will, as Prof. Langley says, rapidly raise to welding-temperature the part directly exposed. But how are shafts going to be welded, or anything larger than the small pieces shown here to-day? Are electric arcs to be spun, as it were, around the mass to be welded, and is the heat to be given in that way, and is it expected that the centre will thus become heated? In that case I do not see why the very fact that Prof. Langley has stated to us of the relative slowness with which heat passes by conduction through the metal would not hinder such a mass from becoming sufficiently heated. If, on the other hand, there is some means by which the heat of the electric arc is to be spread over the ends of the opposite mass, I should like to know the nature of the operation.

MR. WOOD: We had hoped to have had some welding done on a five-horse-power machine, with which we expect to weld two to three-inch bars with equal facility. These samples that I have shown here were welded by one horse-power. As to welding large

shafts, we expect to do it by multiplying the number of arcs in the circuit surrounding the material, and also, if necessary, in front of the faces to be welded, and in that way heat all sides at one and the same time, and perhaps revolving between the arcs to equalize the heat, giving it time to absorb what heat is necessary. I may say, to confirm what Dr. Raymond has said about the material not being damaged, that I have taken a bar of iron and melted it in the presence of the arc drop by drop into a cooling bath. In cooling there would be but a very slight oxidation; and on taking it up it was found to be as soft as the original bar, without any alteration whatever.

AMERICAN BLAST-FURNACE PRACTICE.

[A discussion suggested by the paper of Mr. James Gayley on "The Development of American Blast-Furnaces," read at the New York meeting of the Iron and Steel Institute, October, 1890, and reprinted from the Journal of that Society in *Trans. A. I. M. E.*, xix., 932.]

(Cleveland Meeting, June, 1891.)

[NOTE BY THE SECRETARY.—The three following contributions to this discussion were communicated by letter to the Secretary of the Iron and Steel Institute, and were received by me too late for insertion in vol. xix. of the *Transactions*.]

FRED. W. GORDON, Philadelphia, Pa.: Mr. Gayley's paper calls our attention to the very expensive blunder hitherto made in blowing too much wind in a given size of furnace.

It was quite evident from an analysis of blast-furnace operations that the work we are doing in America is anything but creditable. We were consuming fully 1000 pounds of coke more than is necessary for each ton of iron made, and were hammering the linings of our furnaces out in a very short time, and nothing but the very high price of pig-iron would stand the drain.

It had long been my pet scheme to gradually reduce the quantity of air, at the same time increasing the burden to the maximum and decreasing the limestone, and determining, if possible, the fuel economy which might be reached consistently with the maximum production of the furnace.

After the success of the Isabella furnace at Pittsburgh, which was remodelled in 1880, the North Chicago Rolling Mill Company had

their furnaces re-lined under my instructions, the bosh being decreased from 21 to 20 feet, the hearth increased from 9 to 11 feet, and an angle of 80° to the boshes substituted for one of 75°. The bell was made 11 feet for a top of 15 feet 4 inches. There was then no practical difference between the Isabella and the South Chicago furnaces.

The result at South Chicago fully justified my anticipations. It gave us the starting-point from which to observe what was the most desirable volume of blast. It appeared that this quantity had some relation to the area of the hearth—the most natural thing to base it upon. As, however, there is such a vast difference in the efficiency of blowing-engines, the tightness of the conduits from the engine to the furnace, and the amount of carbon consumed directly by the oxygen of the ores rather than by the blast, the actual revolutions of the engine made but a poor measurement at the best, so that the fuel consumption, which seemed to be a better standard of comparison, was adopted.

During the winter of 1885 and 1886 I was employed as engineer at the Joliet Steel Works, Joliet, Ill., to remodel their furnaces and obtain a fuel economy of less than 2000 pounds of coke to the ton of iron. This was accomplished by re-lining their 20 x 80 ft. furnaces, making the top 16 feet, and putting in 12-ft bells. With these improvements, a product of about 260 tons of iron per day was obtained, equal to 61 cubic feet per ton in 24 hours.

In the following summer the same work was undertaken at the Cleveland Rolling Mill Company's furnace, and 210 tons of iron per day was obtained from a 20 x 75 ft. furnace, on a fuel consumption of less than 2000 pounds per ton of iron. This furnace was altered by simply banking down, and changing the diameter of the bell from 10 feet to 12 feet 6 inches, the top being 16 feet 7 inches.

In 1885 a furnace at Steubenville, O., only 14 feet bosh and 58 feet high, with a 10-ft. top, 7-ft. bell, and 8-ft. hearth, was blown in, and about 650 tons per week were obtained on a fuel consumption of a little over 2000 pounds per ton.

All these furnaces were operated on practically the same materials as those used by Mr. Gayley, namely, Lake ores, or their equivalent in foreign ores, and Connellsville coke.

I was very careful in the above and other similar work to employ the minimum of blast, and to increase it very slowly until the maximum seemed to be reached, or where the driving of the furnace seemed to affect the grade of the iron or the fuel consumption.

Even a marked change in the furnace was not considered sufficient cause to abandon our line of action, for having applied a certain quantity of blast to a certain burden, it was continued until its effect was established in the furnace. It often happened that even with slight changes apparently great disturbances were produced; but they were only temporary, and it was evident that no information would be obtained, and no permanent good results arrived at, if a policy was not followed in spite of all seeming indications that we were in error.

Summing up the result of these and the other operations, it appeared to me that the best measurement for the quantity of blast was the consumption of 5000 pounds of fuel per day per square foot of the sectional area of the hearth.

As these hearths were constructed in very much the same way, having about the same projection and number of tuyeres, it was fair to assume the area for comparison as the area of the brick-work when it was first laid.

Again, at furnace "C" and the Jupiter furnace at St. Louis, operated by the Western Steel Company, these furnaces were blown-in and the same policy was adopted as in the above instances. In this case we had Pilot Knob ore, containing an average of 23 per cent. of silica, of which two-thirds was used, the balance being Iron Mountain washed, a very good and very pure ore, and Southwest, also a fine ore, and working freely in the furnace, the mixture containing from 18 to 19 per cent. of silica. This was smelted partly with Illinois coal in its raw state (as high as one-third by weight was used), and partly with Connellsville, Pocahontas, New River, and Walston cokes. No regular supply of fuel could be had, on account of the disturbed condition of the labor market, and the incapacity of the railroads. Those familiar with the fuels named will note the great variety of their structure and composition, and when we smelted this highly siliceous mixture with these fuels, we were undertaking no small problem. Yet the result was satisfactory, seeing that, from an 18-foot furnace, 75 feet high, we produced 140 tons of No. 1 Bessemer iron per day on a fuel consumption of 2100 pounds; that is, when raw coal was used, estimating a pound of coke as equal to two pounds of coal.

The next experience of importance which I may recite was obtained at the blowing-in of the first furnace at Ensley, Ala. This plant has four furnaces, 20 x 80 feet, hearth 11 feet, top 16 feet, with a 12-ft. bell; angle of bosh, 80°. I gave a detailed account

of the blowing in of this furnace in the *Transactions* of the American Institute of Mining Engineers (vol. xvii., p. 138). The ores contained about 46 per cent. of metallic iron, and the coke consumption per ton of iron was 2240 pounds, with a product of 165 tons per day.

Adding the fuel-consumption of these several furnaces together, with a number of others too tedious to mention, wherein quite a variety of material was used, the average was 4247 pounds per foot of sectional area of the hearth per 24 hours, the extremes being 5200 and 3440 pounds.

If the text of Mr. Gayley's paper is correct in stating that a 22-ft. furnace had an 11-ft. hearth, the coke consumption in his case amounted to about 6000 pounds per square foot, or nearly 50 per cent. more than the above average.

No absolute rule can, of course, be adopted in this, any more than in other furnace practice, though I believe that something near the average might be arrived at by comparisons of a great range of furnaces when doing their very best work, and it seems that it is an important question to consider.

The old question of the temperature of the blast has been very exhaustively discussed from time to time. The inference seems natural that if heat carried in by the blast may be substituted for heat produced in the furnace by fuel combustion up to a certain temperature of blast for a given ore and fuel, each caloric then introduced may supplant that amount of fuel which would have generated a like quantity of heat in the said furnace.

The ratio of CO to CO_2 in the temperature of the escaping gases being the measure of the heat utilization in the furnace, the blast temperature (while within practical limits) should be increased until the percentage of CO_2 falls, or the temperature of the gases unduly increases.

In my practice I have invariably employed the highest blast temperature that could be maintained with a reasonable degree of regularity, and have never found a case where a reduction in the blast temperature would improve the condition of the escaping gases, either in composition or temperature. Hence I believe the limit has not been reached for the highest fuel economy.

If as desirable a gas can be obtained in cold blast-furnace practice as when the blast is heated to 1600° Fahr., and *vice versa*, I think the proposition is proved; but I have found that the lower the blast temperature, the more difficult it was to get, and where

atmospheric temperatures are employed, only selected ores, flux, and fuel will give the result, and that selection has a very narrow range indeed.

A small charcoal furnace was remodelled to my drawings as follows: Total height, 60 feet; diameter of bosh, 7 feet 7 inches; hearth, 3 feet; top, 5 feet; bell, 3 feet; the furnace being blown with one tuyere. There was no semblance of a hot blast. The ore used was a mixture of brown hematite, averaging 50 per cent.; limestone, as pure $CaCO_3$ as could be obtained; the fuel was charcoal, with an average of 85 per cent. of carbon (the balance being mainly moisture). A burden of 1120 pounds of this ore was carried on 600 pounds of charcoal, and a product of 15 tons of Nos. 1 and 2 car-wheel iron per day was obtained.

This result will come out as well as any furnace-work I have any record of, either cold or hot blast, for the utilization of the fuel in the furnace. It was, however, only after repeated trials with many mixtures that such a result was reached. A slight increase of alumina or magnesia, or a change of the character of the ores, though giving almost the same flux, reduced the grade. The furnace could not be safely kept on this burden, while hot blast-furnaces have been kept steadily on a burden equal to this, *plus* that which could be carried by the blast-heat.

Again, though it is heresy to say it, the heat of the blast has no influence on the percentage of silicon. Given an ore mixture, a limestone, and a fuel, and the lowest percentage of silicon that can be reached with cold blast may be slightly reduced by increasing the blast temperature. Not that the blast temperature has any other effect than that the amount of silica present due to the ash of the fuel is reduced by the reduction of the fuel itself to the iron made. It is, however, easier to increase the silicon when using highly heated blast, and only good burdening will keep it under control.

My experience at St. Louis, when smelting the mixture above referred to, carrying 18 per cent. of silica, with New River and Pocohontas coke, containing about 5 per cent. of ash, and only 2 to 2½ per cent. of silica,—having the slag as high as 40 per cent. in silica,—was that the silicon would average too low through the grain of the pig, and the sulphur was all that could be desired in a No. 1 Bessemer iron. The high price of coke (some $6.50 per net ton) made the fuel economy so important a consideration that iron higher in silicon was purchased, while the heavy burdening with the low silicon was continued. Mr. Charles Stewart, general manager, in-

formed me that when the Connellsville coke was obtained in quantity, the silicon was much more at his command. There a small percentage of silica in the fuel governed the silicon in the iron, while the effect of heating the blast some 1600° Fahr. could be neutralized by an increase of burden.

My own and Mr. Edward Dowd's experience at Sheffield, Ala., with Pocahontas coke, has convinced both of us that with pure fuel, ores high in silica can be used to produce No. 1 iron with very low average silicon, no matter how high the temperature of the blast may be.

With these facts, and this sort of reasoning, I fancy that had Mr. Gayley employed 1500° Fahr. instead of 1100°, he might have exceeded his excellent record.

W. HAWDON, Middlesbrough, Eng.: In Mr. Gayley's paper the temperature of the blast is stated to be 1200°, which would appear to many, in England, at any rate, to be low. But as Messrs. Siemens have within the last few years *corrected* their water pyrometer (which is generally taken as the standard in such matters) in such a manner that what formerly was shown as 1500° is now 1250° or thereabouts, this 1200° is probably about the same temperature as was the practice at most of the best English works.

In manufacturing pig-iron in Great Britain, we try to obtain two results, viz., economy of fuel, and, consistently with that, large makes. English practice has been: 1st, economy of fuel; 2d, production. American practice has been: 1st, large production; 2d, fuel-economy. There seems to be an economic limit in driving a furnace which varies, for some hitherto unexplained reason, with different furnaces. Instance the last furnace on Mr. Gayley's list. It is something beyond any chemical phenomena, and appears to be due to such a construction of furnace, and such favorable conditions of working, that the material was presented at the zone of fusion in a manner and condition that caused rapid fusion to take place. And Mr. Gayley seems to have hit this off in the furnace which he has last described. In England we have furnaces very much on the same lines doing excellent work.

Now, the average consumption of coke, leaving out the top item on the list, which appears to be abnormal, is 2400 pounds = 21½ cwt. of coke per ton of iron made. Compare this with English practice, using Spanish ore of 50 per cent. iron, as against the Lake Superior ore of 63 per cent., used at Pittsburgh. It would take at least 1 to 1½ cwt. more coke to smelt this 50 per cent. ore, or, taking

Mr. Gayley's figures, it would take $21\frac{1}{2} + 1 = 22\frac{1}{2}$ cwt. of coke per ton, while in England 19 cwt. and even 18 cwt. is pretty general practice. To make 2000 tons with one furnace in a week requires the same number of blowing-engines, boilers, and stoves as would be needed to make it in two furnaces; and therefore the only extra cost in plant would be, in the latter case, the actual furnace stack itself, costing about £5000, with gallery, etc. By using the two furnaces to make 2000 tons per week, and saving 2*s.* 6*d.* per ton, we should save this amount in 24 weeks. I allowed the $1\frac{1}{2}d.$ extra for additional labor that would be required in the front of the furnace. There would also be a considerable saving in re-lining, which, with rapid driving, is a serious item. Therefore, judging by these figures, it has clearly paid English makers to use two furnaces rather than one.

JOHN M. HARTMAN, Philadelphia, Pa.: I would call attention to the work of a furnace belonging to the Franklin Iron Manufacturing Company of New York, working on fossiliferous hematite of the Clinton group, anthracite coal from Scranton, Pa., and Connellsville coke. The furnace, which has a cubic capacity of 6731 feet, and 10 tuyeres, $4\frac{1}{2}$ inches diameter, 6 feet up from the hearth, has a water-cooled bosh and tuyere and crucible jackets. The hearth is 10 feet 6 inches diameter, and there is one cinder-notch, 4 feet up from the hearth. The furnace is worked with two blowing-engines and three fire-brick stoves. One week's work in 1886 showed the following results:

Fuel, one-third coke, per ton of pig,	2,538 pounds.
Ore per ton of pig,	5,091 "
Limestone per ton of pig,	1,426 "
Blast per minute,	13,514 cubic feet.
Temperature of blast,	1,100° F.
Temperature of escaping gas,	210° F.
Iron produced—No. 1, 342 tons; No. 2, 260 tons; No. 3, 12 tons,	614 tons.
Cinder per ton of pig made,	2,474 pounds.
Yield of ore,	44 per cent.
Ratio of escaping gases by weight $\frac{CO_2}{CO} = .938$.	
Cubic feet of air per minute, per 1000 feet of contents,	2,000 feet.

The furnace, for the whole year 1889, gave the following results after re-lining:

Fuel, one-third anthracite, two-thirds coke,	3,091 pounds.
Ore,	5,376 "
Limestone,	1,485 "

Iron per week,	583 tons.
Output—No. 1, 11,002; No. 2, 16,846; No. 3, 2476,	30,324 "
Average heat of blast,	1,250° F.
Maximum heat of blast,	1,350° F.
Average air per minute,	16,000 cubic feet.
Maximum air per minute,	18,100 "
Average pressure,	9½ pounds.
Maximum pressure,	11 "
Average yield of ore,	41 5/10 "
Air per minute per 1000 cubic feet of contents,	2,277 cubic feet.
Maximum air per minute per 1000 cubic feet of contents,	2,689 "

Discussion at the Cleveland Meeting.

THE SECRETARY: One of our members, Mr. Robert G. Leckie, of Londonderry, Nova Scotia, who is prevented from being here, as he expected, notified me that he should put a question in the discussion at this meeting concerning the method of filling and blowing in pursued by Mr. Gayley and others who had charge of very large furnaces. I advised Mr. Gayley of this, and, being likewise unable to be present, he has sent the following:

JAMES GAYLEY, Braddock, Pa. (communication to the Secretary): The practice that has been followed for a number of years at the Edgar Thomson Works, and which gives excellent results, is practically identical with the plan hereinafter described. As a prime requisite to blowing-in, it is essential to have the furnace thoroughly dried. On this point I would lay great emphasis, as it not only facilitates that operation, but puts the brick-work in a better state for preservation, and conduces to more economical results during the blast. In many cases it eliminates the tendency to scaffolds, which, when formed at this stage, become practically immovable. The drying-out process should be commenced with a very light fire, gradually increased, as the brick-work becomes warmer, to what may be termed hard firing. The furnace should be dried for at least a month, and longer if possible. A statement often made and illustrated in practice, that "blowing-in will dry it out," has resulted in too many cases in loss to the owners. This practice will undoubtedly dry out the furnace, but it is a very expensive way of doing it. In this connection I may cite a case in eastern Pennsylvania, where rival works had furnaces under construction, and each put forth every effort to complete the work first. The successful party in the competition, as soon as the last brick had been laid and the charging apparatus had been placed in position, immediately commenced filling, without removing the masons' scaffolding, and

fired as soon as filled. I have every reason to believe that this furnace was eventually dried out, but the evil results of such haste were discernible throughout the entire period of blast.

In placing the wood in the furnace, our practice is to support on posts a platform about 2 feet above the tuyere-arch, and under the bottom of each post to place a piece of fire-brick, on which is a sheet of thick asbestos. The reason for doing this is, that the furnace interior being very hot, the scaffold is in danger of igniting before the furnace is filled. We usually start to put in the wood in the morning, stopping the firing the evening before, so that the brick-work will be partially cooled. Even then the hearth is sometimes so hot that it is necessary to place a plank on some bats for the workmen to stand on. The advantages derived from this are that the wood, being thoroughly dried and readily ignited, and any excess of moisture being driven off, a quick and easy draft is obtained immediately on lighting, by the natural draft of the furnace alone. After the skeleton parts of the scaffold are in, a charge of coke is put in, sufficient to fill the hearth up to the bottom of the cinder notch opening. This prevents a long drop of the super-incumbent material when the scaffold-posts are burned away, also the tendency to jamb on the bosh, while its commencing to burn as soon as the blast is applied, heats up the hearth and keeps the iron notch in good condition. On the platform, planks are placed sufficiently close to prevent cord-wood on end from falling through. Above the platform, three lengths of cord-wood (hard wood is preferred) are placed on end, with a cribbing in the center, to allow space for the workmen to pass up the wood. On top of the wood a blank charge of coke is put in, of 250 barrows, or, by weight, 207,500 pounds; with this coke there is put in sufficient limestone for fluxing the ash, and in addition a few barrows of spiegel- or ferro-manganese cinder. Our regular charges consist of 12 barrows of coke, 12 barrows of ore, 6 barrows of stone. The weight of a barrowful of coke is 830 pounds; the regular coke charge amounts, therefore, to 9960 pounds. With the large blank charge of coke that is put in, the ore-round can be started at about the same weight as the unit of coke; consequently we put in the first charges as follows:

No. of Charges.	Weight of Coke.	Weight of Ore.	Limestone.
10	9960	9,600	
5	9960	10,200	to
10	9960	10,800	suit.
20	9960	12,000	
30	9960	12,600	

We charge afterwards as the condition of the furnace indicates. Of course it only requires a small number of the above charges to fill the furnace. To the first few charges an extra barrow of furnace-cinder is often added.

The space between the scaffold above and bed of coke beneath is then filled with the kindling-wood, and the torch which is to start the slumbering energies into activity is applied, generally by a young lady, whose graces preside as beneficent genii over the destiny of the blast. In addition to lighting the wood at the cinder notch, a red-hot bar is thrust in at each tuyere to start the combustion uniformly. Ordinarily we find very little trouble with the draft, and when properly managed I consider the natural draft of the stack much preferable to the more rapid one obtained by connecting with the chimney. When the scaffold has burned away, allowing the stock to settle gently, and bringing hot coke or charcoal in front of all the tuyeres, the blast is put on. The time from lighting to putting on of blast varies from 6 to 10 hours. The blast is put on slowly at first, and increased hourly until the volume of air is one-half the normal quantity, at which point it is held until after the first cast of iron is made. In order to avoid explosions in the flues, which frequently happen at the start, the valves in the boiler and stove-gas mains are closed, and all of the gas is allowed to escape through the bleeder until after the first cast is made.

A practice in frequent use for blowing-in furnaces, is to put in a large quantity of wood with a small blank fuel charge; this being followed by the regular charges, in which the ore weight is about one-half that of the fuel. The result of this practice is that the wood burns away quickly, bringing the ore down to the tuyeres before it has had time be reduced; making a cold hearth, from which is got a black scouring cinder and white iron. The method that I have described in detail has much to commend it, as the large blank charge of coke used furnishes sufficient gas and solid fuel for complete reduction, and after settling, when the wood is burned away, maintains the fusion limit at or near the position that would be normal for that volume of air. At the same time the hearth is thoroughly heated, and a good grade of iron is obtained. As a matter of fact, the first cast of iron is No. 1, varying in silicon from 1.75 to 2.50 per cent., and the cinder is identical in color and composition with that obtained when the furnace is in normal working condition. So certain are we of knowing by this practice the quality of the metal, that it is run into ladles and sent to the steel-works for conversion like the

rest of the metal. The description I have given of our practice of blowing-in is based on a furnace of 19,000 cubic feet capacity.

J. E. JOHNSON, Longdale, Va.: In confirmation of the remarks of Mr. Gayley as to the imprudence of too little drying-out, I can give an experience in point. We re-built an oven, and the time consumed was about two months. The result was that while the furnace started very quietly and without any scaffolding, it was six months before it carried the proper burden. She was dried for only two weeks before we put the blast on. The great mass of brick-work was undoubtedly still filled with moisture, and the result was that we had a very poor blast, only about two-thirds burden being carried, and the iron being very unsatisfactory. An incident of the starting of that blast was, that instead of beginning on Saturday, as had been intended, we waited until Monday morning. On Sunday there came a very heavy rain. We did not have any stock-house; we kept all our material out of doors, and everything was wet except the wood. We thought perhaps we might dry out the material a little by blowing a hot blast in after the furnace was filled, and before kindling. We got it a little too hot, and the wood kindled, and we started the furnace with a hot blast. Fortunately we had no evil result.

HUGH KENNEDY, Etna, Pa.: In blowing-in our furnaces we very seldom put in more wood than one layer above the tuyeres. We put in a large amount of coke, probably twenty rounds, at 4000 pounds per round, and enough lime to flux the silica in the coke. Then the furnace is filled to about even quantities of ore and coke. After attaching a suitable arrangement to the tuyere-pipe, we pour a bucket of petroleum into the furnace at each tuyere before lighting. Then we put on a light blast, and light at the tuyeres, and keep the blast on the furnace. In that way the fire is distributed equally through the wood, and the coke comes down to the tuyeres pretty soon. Then we close our bell, and bring our gas down to our stove and boilers. We have found that with a large amount of wood put in the bottom of the furnace, leaving the top of the furnace open and putting on the blast, so much settling leaves a space in the top of our furnace which is apt to make a bad explosion; and we think it preferable to put in a small amount of wood and leave little chance for settling for some time after putting on blast. In the matter of drying-out our furnace, we have dried out for two weeks and for three months, and ordinarily we have seen very little difference in the working of the furnace, provided we gave enough coke in our charge not to bring our ore down our tuyeres too soon.

PRESIDENT JOHN BIRKINBINE, Philadelphia, Pa.: The practice described by Mr. Kennedy, of filling the furnace, putting on the blast, and then lighting it, is quite different from that followed in some of the older charcoal-furnaces, according to which the furnace was filled with charcoal and lit on top, and when the fire had worked down to the tuyeres, the manager would fill the furnace again and then put the blast on. I remember some years ago visiting one of these furnaces, and remarking to the proprietor: "I see you are putting your furnace in blast; and still light on the top." He replied, "Yes; but yesterday I thought she was not burning fast enough, so I fired her at the bottom, too."

Beyond question success largely depends upon blowing-in. Sometimes the preliminaries for a good start are carried to such a length of time as to spoil it. Within a year I visited a plant where the furnace had been filled and fired, the fire having been allowed to creep along for two days and a half and then blown very gently. It was five and a half days from the time the fires were lighted before any pig-iron was cast; and the furnace had a very short life. After the fuel in the furnace was kindled the superintendent was prepared to begin blowing at eight in the evening; but arrangements for a formal opening had been made; and the gentleman who was to make the speech and the minister who was to make the prayer were not there, so matters were left until eleven o'clock the next day before the blast was put on.

E. C. PECHIN, Roanoke, Va.: Does Mr. Kennedy, in filling, dump his coke-charges from the top of the furnace, or are they lowered? [Mr. Kennedy replied that they were dumped.] I am inclined to think that with our Pocahontas coke, which is a very much softer material, that practice might be accompanied with a little risk. I should think it better to lower the blank charges. In our practice we put on the light blast first before we fire up, and then, just as soon as the gas will ignite freely, shut the bell. It only shows how doctors will differ. In many places they insist on the old way. I think the gentlemen who were down at Birmingham know something about it. We can recollect a furnace there that had been started after the old anthracite practice, and they were in a pretty bad scrape the morning we were there three years ago. The condition of the tuyeres was very ugly, and I asked them when they had lighted up? They had lighted up, it appeared, without having all the connections made, and with some of the necessary parts missing at the tuyeres; and they had had the fire on, without any blast, for

thirty-six hours. I think I had rather take my chances on blowing-in with the blast.

MR. JOHNSON: I would remark that we dump right into the furnace in filling. Our height is 60 feet.

MR. PECHIN: But the difference between 60 feet and 75 feet may be a good deal with Pocahontas coke.

GEORGE A. DEAN, Mingo Junction, Ohio: I would like to hear some opinions as to the effect on a furnace of standing after having been relined for any length of time before being fired. Is it objectionable, or not? I had an experience myself with a furnace that we had lined up that stood about seven months, and we kept it warm by putting through it the gas coming from another furnace in operation. The subsequent blast only lasted about half the normal period before the lining was worn out, and the question in my mind is, whether the gas from the other furnace had in any way affected the brick-work of the lining.

A MEMBER: Was that furnace roofed over the top?

MR. DEAN: It was not perfectly tight, but it had roof enough to shed a heavy rain.

JULIAN KENNEDY, Pittsburgh, Pa.: As to blowing-in, it is better to be too hot than to get too cold. My plan is always to put in one layer of wood above the tuyeres, and then go on up with a ring about a foot thick. Then, as Mr. Gayley says, put in a charge of coke, and then with pretty heavy charges in that way you get a good hot fire. I myself was the first person, I think, who ever blew in a furnace, in which the blast was put on before lighting. That was the B furnace of the Edgar Thomson works. I have never heard of its being done before. The way we blew in when I first commenced to learn the business was to put in the wood and fill in 10 or 12 feet of coke, and light that below and wait until that burned through; then fill another layer of, say, ten feet, and wait again till the fire showed on top, and so on till the furnace was full. We always knew when it got through by a heavy explosion. I think that the practice of leaving the bell open after the blast is on is a bad one. I think that as soon as the blast has been on for a couple of hours, and you get the gas in shape so that it lights easily, the bell should be closed.

As to the question of the effect on a furnace of standing, I think the greatest danger of injury is from rain getting down between the lining and the casing.

Where you have several inches of loam back of the brick a great

deal of moisture may creep down there, unless it is very carefully protected. Even if you keep a fire inside the furnace, a constant stream of water going down outside will disintegrate the brick very rapidly. If Mr. Dean's furnace stood idle in the winter, the brick may have frozen.

MR. DEAN: We kept the furnace warm all the time with the gas from the other furnace. The furnace was hot below and the bell was warm. I may add that the furnace was lined in the fall, and was thoroughly dried out, and that the gas was kept on and the furnace kept hot during the whole interval; there was not a time from the fall until the next spring, when we blew in, that the furnace was not hot—the bottom of it was red-hot, from heating with gas through an eight-inch pipe. But the result of the blast was so unsatisfactory that the question has always been in my mind whether it is as well to line a furnace before you want to use it, as to line it just in time to dry it out. We have two furnaces at our place, and we have an equipment for running both. Our policy is, however, to run one of the furnaces at a time, and by the time the life is out of the furnace reline it and use the other furnace. In this case we lined up the furnace about nine months before we wanted to use it.

MR. H. KENNEDY: My experience has been that if the fire-brick have been frozen you cannot expect them to last well in the furnace lining. As the last outside layer of the lining would probably be the part of the lining to serve for the last half of the campaign, I think such a defect in that part would undoubtedly account for Mr. Dean's lining not doing good work. We had a lining got ready probably two years ago, and, for lack of better storage-room, we piled it up and roofed it in carefully, to keep it dry. There came a big flood in November, and the water came up six or eight inches on those bricks; and after the water went down the bricks were frozen in ice all winter. The question came up what would be the effect on them. They were a part of the top brick. We concluded to put in the lining just as it was and run the risk. A little less than a month after the furnace had been put in blast I cut a hole in under the top and found the brick-work just like a pile of ashes. That lining made 166,000 tons; but when we came to blow out we could find holes in the lining where the brick-work was all gone, showing to my mind that the freezing had destroyed the brick and made them disintegrate under heat. I have also seen furnace-linings that have stood all winter and been frozen, and my theory is that no matter how you dry such a furnace-

lining and then blow in, four or five days after you have blown in, the water will run down in streams out of the brick-work. You have got to get in heat enough to drive it out. To stand over winter I think will ruin any lining.

MR. PECHIN: I am thoroughly a believer in what Mr. Hugh Kennedy has just advanced; and the point that interests me greatly, and to which I pay the largest attention in specifications in the building of a blast-furnace, is that every fire-brick that is to be used in the furnace must come under cover and be kept under cover at all hazards. I believe the largest damage that can come to a furnace-lining is by reason of the disintegration of the bricks, wet before putting in, or improperly dried. In Virginia there has been very serious trouble by reason of the giving way of the in-walls immediately under the bell. A furnace, built last year, only ran about six or eight months, when one morning the whole of the lining under the bell gave way, and the destruction followed right down, and they had to shovel out. The bricks were absolutely disintegrated. I think unquestionably that disintegration took place by the enormous amount of hot vapor that was generated in the upper 12 feet of the furnace. They had used No. 2 Mount Savage brick, which I think is bad practice in that part of a furnace. The brown ores have from 6 to 12 per cent. of moisture; the Pocahontas coke, exposed to rain, absorbs much water; and the result is that there is a perfect current of hot vapor passing off in the top of the furnace, and I am inclined to think that that peculiar condition results in wasting the brick. Therefore, I have strongly recommended in our Virginia furnaces—as I would for any furnace, in fact—the putting in for the upper 15 feet of the furnace as nearly a vitrified brick as can be got.

THE PRESIDENT read the following communications, giving data of practice at American furnaces:

R. W. LEE, JR., Logan, Pa.: The Emma furnace, of the Logan Iron and Steel Co., is an old structure which has been raised, its dimensions being as follows:

	Feet.
Total height,	52
Diameter at top of bosh,	9
Diameter of hearth,	5
Diameter at stock line,	7
Height of tuyeres, above bottom,	5
Total cubic capacity of furnace,	1449 cubic ft.
Cubic capacity to stock-line,	1303 "

The furnace is blown by one vertical Weimer engine, with 14-inch

steam cylinder, 48-inch air cylinder, stroke, 2 feet, running at an average speed of 80 revolutions per minute, and this engine has been in use since July, 1881, running continuously at from 70 to 90 revolutions per minute; the only repairs being required were the insertion of some new rubber blowing-valves and renewal of crank-pin brasses. The air is heated in one Durham stove up to 800° Fahr.

The furnace is running on an ore-mixture of soft fossil- and roasted carbonate-ores, from the company's mines, mill-cinder and some Lake Superior red hematite; the mixtures vary according to the uses to which the iron is to be put, the percentage yield of the ore-mixture running from 57 to 43 per cent., and the average being 47 per cent. During the present campaign, which commenced June 12, 1890, the furnace has averaged, without making any allowance for stops, 24.2 gross tons per day, the maximum output, which has been repeated a number of times, being 30 gross tons.

The operation of the furnace showed a consumption of 2900 pounds of coke per ton of iron; the ores charged averaged 4873 pounds per ton of iron made, and in addition 2561 pounds of limestone were charged. This is equivalent to 2.56 pounds of ore and flux per pound of fuel; and it is proper here to state that, owing to the fuel-record being charged with a percentage-allowance for loss, the furnace-bin shows a considerable stock on hand which does not exist on the books; therefore, the actual fuel-consumption is somewhat less than above, and the amount of material carried per pound of fuel correspondingly greater.

When running at 30 tons per day, 1 ton of iron was made in 24 hours for every 43.4 cubic feet of furnace-capacity; and when running at the average of 24.2 tons per day, 1 ton of iron was made for every 53.8 cubic feet.

EDWARD DOWD, Sheffield, Ala.: The Hattie furnace is one of the two furnaces at Sheffield, Alabama, owned by the Lady Ensley Coal, Iron and R.R. Co. It has the following dimensions:

Total height,	75 feet.
Height of tuyere-line,	5½ "
Height of bosh,	35 "
Diameter of hearth,	9 "
Diameter of bosh,	17 "
Diameter of stock-line,	12½ "
Diameter of bell,	8½ "
Cubical capacity,	10,000 "
Number of tuyeres,	8
Size of tuyeres,	6 inches.

Volume of blast per minute,	14,000 cubic feet.
Temperature of blast,	1100° Fah.
Pressure of blast,	5 lbs.
Average make of iron for six months ending March 30, 1891, per day,	146 tons of 2300 lbs.
Average consumption of coke per ton of iron,	2451 lbs.
Average yield of ores,	48½ per cent.
Ore used, native brown hematite.	
Coke used, Pocahontas Flat Top.	
Limestone used, Darlington Oolitic.	

Tabulating this information and comparing it with the best work reported by Mr. James Gayley in his admirable paper, read at the American meeting of the Iron and Steel Institute, on "The Development of American Blast-Furnaces, with Special Reference to Large Yields," we will compare with the furnace doing the best work, and described in his paper in Fig. 8, being one of the Edgar Thomson furnaces.

	Fig. 8 of Mr. Gayley's Paper.	Hattie Ensley Furnace.
Cubical capacity, cubic feet,	18,200	10,000
Volume of air per minute, cubic feet,	25,000	14,000
Temperature of blast, degrees Fahr.,	1,100	1,100
Pressure of blast, lbs. per sq. inch,	9½	5
Number of tuyeres,	7	8
Size of tuyeres, inches,	6	6
Average daily output, gross tons,	310	146
Average coke consumption, lbs. per ton of iron,	1,920	2,451
Furnace-capacity for 1 ton of iron,	59	68
Average yield of ores, per cent.,	62	48½

To arrive at a fair comparison we must take into account the difference in the size of the two furnaces, and the difference in the yield of the ores.

Bringing the Hattie up to the same size as Mr. Gayley's furnace Fig. 8, and ores up to the yield of his own, we have the following, showing more nearly a true comparison.

	Fig. 8 of Mr. Gayley's Paper.	Hattie brought to size of and ore of Fig. 8.
Cubical capacity,	18,200	18,200
Volume of air per minute,	25,000	25,500
Temperature of blast,	1,100	1,100
Pressure of blast,	9½	5
Number of tuyeres,	7	8
Size of tuyeres,	6	6
Average daily output,	310	340
Average coke-consumption,	1,920	1,917
Furnace capacity per ton of iron,	59	54
Average yield of ores,	62	62

It will be seen that the small furnace is producing at a higher rate than the large one, when the compensations are taken into account, and on as low a consumption of fuel, notwithstanding that the small furnace, by reason of having more tuyeres and as many coolers, suffers a loss of almost double the amount of heat per ton of iron made, in the tuyeres and cooler-water.

THE PRESIDENT also read the following statement, furnished by Mr. E. Fitz Gerald, manager of the Antrim Iron Co., Mancelona, Mich.:

On blast No. 2 we blew in on the 1st of June. The first year we ran 340 days; product, 16,152 tons; the second year, 352 days; product, 16,368 tons; the third year, 361 days; product, 19,561 tons; the fourth year, 340 days; product, 22,412 tons; the fifth year, 345 days; product, 25,563 tons. For 8 months of the last year No. 1 stack also was in blast; product, 15,062 tons, making a total of 40,625 tons for the fifth year. The first blast of the original stack covered a period of 22 months; product, 30,283 tons.

To-day the only parts of the original purchase left are the engine and oven. All the buildings have been removed and replaced and added to with brick and iron. Two new engines (30 x 48 x 72), No. 1 and No. 2 stacks, 3 ovens, 11 boilers, new hoisting engine, angle-iron tower, iron gallery and platforms, 4 pumps, 93 kilns, office, store, dwellings, warehouse, shops, stables, etc., have been erected.

From the date of the purchase, March 5, 1886, to the present time, the property has been idle 148 days. Total product to date, 115,118 tons.

Antrim Stack, No. 1. General Dimensions.

Total height,	48 feet.
Diameter of bosh,	10 "
" crucible,	5 " 6 in.
" tunnel-head,	7 "
" stock-line,	8 " 6 in.
Gross cubic contents,	2415 cubic ft.
Net cubic contents,	2176 "
6 tuyeres, diameter,	4 inches.

Performance, Blast No. 1, April 14, 1888, to April 22, 1891.

Total number of days in blast,	1,103
Days in actual operation,	1,057
Small Weimer engine (16 x 30 x 48), in service,	582 days.
Product,	32,326 tons.
Daily average,	56 "

Large Weimer engine (28 x 48 x 72), in service,	475 days.
Product,	34,584 tons.
Daily average,	73 "
Total product of blast,	66,910 "
Daily average,	63 "
Charcoal used,	6,811,700 bushels.
Charcoal per ton of iron,	103 "
Yield of charcoal per cord of wood,	47 "
Ore used,	116,380 tons.
Average yield in furnace,	57½ per cent.
Average burden per charge,	1,340 lbs.
Pounds of ore charged per pound of coal,	2.07
Air-discharge, per revolution, small engine,	62 cubic ft.
Average revolutions per minute, small engine,	75
Air-discharge, per revolution, large engine,	226 cubic ft.
Average revolutions per minute, large engine,	35
Largest day's product,	93 tons.
Largest week's product,	580 "
Largest month's product,	2,410 "

FAYETTE BROWN, Cleveland, O.: We have at the works of Brown, Bonnell & Co., Youngstown, O., two furnaces, one of which is probably the smallest furnace in the State. One is 55 feet high, 12½ feet bosh. The other is 14 feet bosh and 60 feet high. They are old-fashioned shells, very small at the top, very little room in the shell. Each of them has two small iron stoves. The small one runs frequently 100 and sometimes 110 tons a day, generally about 90 tons. The other one runs from 110 to 138 tons. I think the two furnaces will have made about 70,000 tons of iron in the year ending the 1st of July, 1891. The consumption of coke is larger, of course, than it is in some of these large furnaces, for the reason that we use a great deal of lime,—some of the time as much as 1500 pounds of lime to the ton of iron. Our coke runs up then to about 2600 pounds to the ton. When we use less lime, we get the coke down to about 2300 pounds to the ton. The furnaces are old-fashioned. One of them has only one engine, 84 inches by 4 feet stroke. The other has an engine of the same size and another of a small size. The furnaces are anything but models, but they have done good work. The tons of pig-iron referred to are gross tons, of 2268 pounds if cast in the sand bed, or 2240 if cast in chills. The tons of coke, coal and limestone are net tons of 2000 pounds.

MR. H. KENNEDY: The Isabella furnace No. 2 was in blast 4 years 11 months and 5 days on mill-iron continuously, using never less than 25 per cent. mill-cinder and coke, which will run 15 per cent. ash, and made in that time 297,968 gross tons of iron, with 2350

pounds of coke to the ton. It was lined to 18 feet bosh, with 14 feet stock-line and 11 feet hearth.

MR. JOHNSON: These immense furnaces that Mr. Gayley writes about seem to overwhelm our little furnace. We make only about 100 tons a day in two furnaces, one of them 14 and the other 16 feet in diameter, and each about 60 feet high, with 10-foot tops. No. 2 does rather better than No. 1. It is making 55 to 60 tons a day, with a consumption of 1900 pounds of coke. Our tons are 2280 pounds, including the allowance for sandage. We have done a great deal better than that in times past. As long ago as 1879 we ran a whole year with a small furnace 11 feet in diameter, 60 feet high, and 8 feet top, where the year's work showed about 2000 pounds of coke to a ton of iron, and one week's record was 219.5 tons of iron to 187 of coke; but for some reason we are not able to do that any more. The ore now yields about 45 per cent., as against 47 formerly. Of course, the additional 2 per cent. in the ore helped us very much in fuel-economy.

LEE BURT, Detroit, Mich.: For the benefit of some gentlemen in the charcoal-iron business, I will say, as a director of the Union Iron Co.'s furnace in Detroit, that during the last year our average make was 32.9 tons (at 2240 lbs.) of iron per day, with an average fuel-consumption of 87.11 bushels (20 pounds each), or 1742 pounds, per ton. It is a small furnace, built in 1872, and the lines have not been changed since 1879, except to increase the height of the tuyeres from 3 ft. to 3 feet 8 ins., leaving 9 ft. 4 ins. from center of tuyere to top of bosh. The total length of the last blast was 962 days; product of iron, 31,652½ gross tons; consumption of charcoal, 2,759,539 bushels; of ore, 53,407 tons (yield, 59.2 per cent.); limestone, 1505 tons (2.8 per cent.). Yield of charcoal per cord of wood, about 49 bushels. The heat of the blast was about 800° F.

MR. J. KENNEDY: I would like to hear from some of our mathematical friends as to comparisons between furnaces of different sizes, and also with different ores. My experience has been that, on a 58 per cent. mixture, I could always get better yield, in proportion, than on a higher mixture. Some ores, again, will give a better yield than others which carry no lime. I have always had the impression, also, that a furnace of 9000 cubic feet capacity would make more than half the iron that one of 18,000 cubic feet capacity would.

R. W. RAYMOND, New York City: As far as our experience at the Durham furnace shows,—and I suppose on this point an anthra-

cite-furnace may be as good a source of information as a coke-furnace,—we find, first, that we cannot make comparisons based solely upon the percentage of iron yielded by the ore. Of course it is an essential matter what else is in the ore besides the ore which yields the iron; for instance, whether it is a self-fluxing ore, or even one which carries so much lime that it can flux other ores. These elements directly determine the weight of the furnace-burden, and to determine what Mr. Hartman calls the carbon duty of the fuel consumed in your furnace, you must take into consideration both the weight and the nature of the burden; the amount of ash in the fuel; the amount of slag made; the heat-absorbing capacity of that particular slag; the grade of iron produced, and the heat-absorbing capacity of that grade. The comparison thus becomes very complicated. Secondly, as to comparisons directly calculated from cubic capacity, I think that Mr. Kennedy is perfectly right; and perhaps the reason is, that merely doubling the capacity of a stack is not necessarily doubling its productive elements. There is, in the first place, an unknown excess of space in the reducing zone in the upper part of the furnace. I think this is generally true of American blast-furnaces. We are on the safe side, and, as a general rule, I do not think our furnaces, when running normally, whether on the hematites of Lake Superior, brown hematites, or magnetites, permit unreduced ore to reach the tuyeres through lack of reducing-space. Now the height (*i.e.*, the vertical thickness) of the melting-zone does not increase in proportion to increased dimensions. Consequently, if of two furnaces, *exactly similar in form*, one has four times the capacity of the other, its melting-zone will have little more than double capacity. This is an extreme example; and, moreover, large and small furnaces are not usually similar in form. But the illustration shows that aggregate capacity is no basis for direct comparison, because it does not furnish the necessary data concerning the actually productive part of the furnace. In other words, the unknown excess of reducing-space is probably greater in proportion in large furnaces than it is in small ones, and hence smaller furnaces are likely to make more iron in proportion to their aggregate cubic capacity, because there is relatively a larger part of the furnace directly producing, and a smaller part inactive. Mr. J. S. Fackenthal, superintendent at Glendon, tells me that the smaller furnace there produces more in proportion, and I presume the experience is general.

Again if it be measurably true that increased productive capacity

must be measured by the capacity of the melting-zone, and that this zone cannot be increased to any great extent in vertical thickness, it follows that the increase must be mainly in diameter. And here we are limited by the conditions determining the penetration and amount of effective blast. A large furnace with inadequate blowing-capacity or unsuitable tuyere-arrangements cannot be driven to its full capacity. And there is a point where increase of diameter at the tuyeres ceases to be beneficial anyhow. Moreover, putting in more air, after the fuel in the melting-zone has received all it can utilize in the unit of time, is worse than useless. In our anthracite-furnaces we reach the limit sooner than with coke; and this is the principal reason of the present general practice of mixing coke with the anthracite (generally, I think, about 20 to 25 per cent. of coke), in order to get a fuel which will take more air within the space of the proper zone. Now, all these elements enter into the calculation of the relative efficiency of furnaces, and I very heartily second what I suppose to be the underlying thought of Mr. Kennedy's remark. I do not think the hypothetical comparison made by Mr. Dowd valid. The coal we use is lump-coal—the largest size. Our experience is that somewhat smaller sizes are just as good.

ALEXANDER E. BROWN, Cleveland, O.: Is not the actual theory and fact that the smaller coal, say egg size, is much better than anything larger—that is, arguing upon the same theory that the coke is porous and has a large surface immediately exposed to the air, and from the statement that the anthracite will only receive so much air at a time, if you break up that lump that is only exposed on the outside you have quadrupled the surface? I know in one furnace just simply the suggestion of that increased the output nearly 50 per cent. using small coal, and breaking up the limestone in equal proportion.

MR. RAYMOND: It must be remembered that lump-coal is cheaper than the prepared sizes, and I am inclined to think the coal is considerably broken up by the time it gets down into the zone of the tuyeres to burn. On the other hand, it is not certain that uniformly small coal will, by reason of its greater exposure of surface, act like porous coke. It may pack and resist the passage of air. We are all sometimes troubled with high pressures. We are blowing a great many more pounds with our engines at most American furnaces than used to be considered at all wise; and the increased pressure of blast which might be required to force air through fine material is a pretty serious question, when our margin of pressure is

small anyhow. We are taking in a great deal more fine ore than formerly. There is a considerable percentage of fine ore used in the Durham furnace; and all through our part of the country our blast-furnace managers are nerving themselves to use, not merely the amount of fine ore that they make in handling their own ores, but also other fine ores, such as concentrates, or ores like those of the Belvidere mine in New Jersey, which are very rich, but almost like sand.

MR. A. BROWN: Is it not a fact that the amount of air taken into the furnace per ton of iron produced is in inverse ratio to the surface exposed by the fuel? Charcoal, which gives by far the largest exposure of available carbon, takes the least of all fuels in air per ton of iron and coke next, while anthracite stands highest in the requirement of air, and the coarser the coal the larger the volume of air required. If the passage of air is delayed by the fineness of the fuel, it is delayed to burn, and cannot go through to be wasted.

MR. BIRKINBINE: Mr. Brown's original statement, that the smaller the coal the better work you get out of a blast-furnace, agrees with my views. My reason for saying that is that, some years ago, I was commissioned by one of our large coal and iron companies to make some investigations as to the possibility of the further extension of the use of anthracite coal alone in the manufacture of pig-iron. In doing so I found that in every case investigated, where a blast-furnace had been forced by reason of strikes, when they could not get lump-coal, to use the smaller sizes, the furnace made more iron in the same length of time and with less consumption of fuel. It is economically, however, an important fact that the egg-coal is more expensive than the lump.

The results of experiments in the use of magnetic concentrates in the anthracite furnace at Port Henry will, I trust, be fully communicated to the Institute hereafter by Mr. Langdon. Meanwhile, I can say, in a general way, that the use of this material considerably increased the daily yield of the furnace, while it decreased the amount of limestone used and fuel consumed; but it did not increase the proportion of foundry-iron made. About 1400 tons of concentrates were thus used, in the proportion of about 40 per cent. of the total ore-charge. This was subsequently carried to 60 per cent. and more, with a further decrease in fuel-consumption.

MR. DEAN: The furnaces that we operated first were 15 feet in diameter and 60 feet high. At that time we made 80 tons of iron a day. We afterwards changed our furnaces to 17 feet diameter and

75 feet high, which is about 50 per cent. more cubic capacity, and we made 135 tons a day, with a blast temperature of 800 degrees. The increase seemed to have kept up the ratio. The difference that we found in the furnaces was that the smaller furnace, 15 x 60, under the same management, on the same ores, used 2800 pounds of coke per ton of iron, while the larger furnace used but 2400. By getting increased heat we have been able to make over 200 tons a day on less than 1900 pounds of coke, with a temperature of 1300 degrees. Our experience seems to prove that we have increased our product in proportion to the number of cubic feet in the furnace.

MR. PECHIN: This matter is getting pretty deep for an off-hand discussion. There have been at least half-a-dozen points brought up here in the last half-hour, each of which is of sufficient importance to keep us all here for a pretty long session. It seems to me that in endeavoring to compare the work of different furnaces, you must take one thing into very careful consideration—not merely the percentage of iron in the ores, but the shape in which that percentage exists. Mr. Kennedy will recollect that last fall, in New York, when some of our English friends were making herculean attempts to show that we ought not to get such big yields out of our furnaces as we did, and that there was no good reason why we should, I asked him, after one of the sessions, this question: Whether the physical conditions of the ore had not a great deal to do with the large yield? I had asked Sir Lowthian Bell that question before, because it seemed to me that I recollected something in his book that had a direct bearing on the subject. But he was not disposed to concede that it made any difference in the present case. Mr. Kennedy, however, told me that if, in place of the soft Lake Superior hematites, used by Mr. Gayley and himself, they had a hard specular Lake Superior ore, they would not have got anything like so large a yield. If, in place of the hard ores of Lake Superior, we should put on the magnetic ores of Lake Champlain or New Jersey, we would find a very marked difference again. In his book, as I recall it, Sir Lowthian Bell makes a special point as to the rapidity of reduction owing to the different physical conditions of the ores—that the Cumberland hematites reduced in a given time; the Cleveland stone took much longer; the specular ore took a very long time. He makes another point there, which seems to me to have some bearing on the inquiry as to the use of fine concentrates. I think he says that, so far as Cleveland iron-stone is concerned, reduction practically took place within 12 to 15 feet of the top of the furnace; and that at that poin$_t$

he found no ore whatever except in the shape of powder. And if the reduction takes place in proportion to the rapidity and the readiness with which each particle of ore comes in contact with the volume of carbonic oxide gas going up through the furnace, why isn't it a great deal better for rapid reduction that the ore should go into the furnace small? Do not lumps have to be broken up by decrepitation in the furnace before they are in condition for carbonic oxide gas to act readily upon them in reducing them? As to Mr. Dowd's comparison, he has extraordinary material to work with. He has a beautiful brown ore, as clean as any I ever saw in my life. I have gone into his stock-house and taken up a handful of brown ore, which I could have put into a white pocket handkerchief without soiling it, so far as any accompanying clay or earth was concerned. Every little particle of it is hard and separate from each other. That beautiful stuff goes into the top of his furnace in a splendid condition. It is a porous ore. The result is that he can drive the furnace for all it is worth, and he does get an extraordinary yield with Pocahontas coke. But, if he were to put our wet and soggy mud into his furnace, I doubt whether he would be able to get up a paper of comparison such as has been presented to us to-night.

I am perfectly willing to concede that the percentage of iron in the ore is an important factor; but the question seems to be quite as important, whether the ore is reducible more or less easily, and whether its mechanical condition is favorable to rapid reduction.

[The following is a continuation of Mr. Pechin's remarks in a subsequent communication to the Secretary.]

Since the above discussion took place I have taken the opportunity of looking up Sir Lowthian Bell's experiments, and they seem to sustain the position, that the physical condition of the ore has much to do with the rapidity of reduction, and, consequently, working.

On page 18 of *Chemical Phenomena of Iron Smelting*, after giving the experiments made by him, Nos. 31 to 41, he says: "These experiments are sufficiently instructive in pointing out the influence which molecular structure is considered to exert in rapidity of deoxidation." In continuing his experiments Sir Lowthian found (page 19) that reduction was greatly accelerated by passing a rapid current instead of a slow current of CO over the ore, and that this was particularly marked in the Lancashire hematites as against the Cleveland carbonates and the Elba speculars.

Compared with each other, the percentage of original oxygen lost under the conditions of his experiments (page 19) was as follows:

	Lancashire.	Cleveland.	Elbe.
Slow current of CO, ...	35.9	37.3	16.8
Rapid " "	70.9	50.6	18.1

It is to be regretted that Mr. Gayley in his experiments (*Trans.*, xix., 991) did not also try the effect of a rapid, as well as a slow, current on the Lake Superior ores. Such a test might have given valuable information. It seems safe to assume, however, that, with our rapid driving, our soft hematite ore reduced with extraordinary rapidity, and reaching a greatly enlarged zone of fusion in a prime condition, enormous quantities are almost instantly melted, resulting in our large outputs.

At any rate, we know by experience that in the same furnace, when we use a less reducible ore—even if richer in iron—the output is greatly decreased.

Does it not seem to follow that "molecular structure" may have a good deal to do with it?

MR. RAYMOND: Let me add that the paper of Mr. Jasper Whiting, "Some Experiments on Blast-Furnace Gases," presented at this meeting, illustrates very forcibly the complexity of the problem we have been talking about; and also the true, though most patient and laborious method of dealing with it. The diagrams of Mr. Whiting will be engraved, and members will find the paper worthy of much deeper study than can be given to it here. After all, the blast-furnace gases tell the story of technical economy.

MR. PECHIN: It is of course impossible to discuss at the moment such a paper as Mr. Whiting's. It strikes me as one of the most valuable ever presented to the Institute.

SOME EXPERIMENTS ON BLAST-FURNACE GASES.

BY JASPER WHITING, SOUTH CHICAGO, ILL.

(Cleveland Meeting, June, 1891.)

IT is well known that there is no better indication of the working of a blast-furnace than is given by analyses of its gases; yet although many individual analyses have been made, there is, I think, no

record of any continuous series of such analyses extending over a considerable length of time.

With the hope of obtaining a clearer knowledge of the influence which the different factors in furnace-work exert on the composition of these gases, such a series was undertaken. Three furnaces belonging to the Illinois Steel Company were chosen for experiment, because, while they were being worked under quite similar conditions in regard to ores, fuels and fluxes, they varied largely in shape, size and rate of driving. Four to six analyses were made each day, and, with each analysis, observations of the several factors were taken. These observations, together with the ratio of CO_2 to CO in the gases, are tabulated in graphical form on the accompanying diagrams.

Before examining these diagrams, it may be interesting to glance at the changes which the composition of the gas underwent during the "blowing-in" of Furnace No. 1, Union Works. It was lighted at 10 P.M., August 8, 1890. Just ten hours later the first sample of the gas was taken and found to contain CO_2 and CO in the proportion of 1 to 2.3. Gradually this increased until at 1 P.M., the hour at which the blast was put on, it became 9.1. From this time, as the furnace grew hotter, this ratio very rapidly increased, and at 3.30 P.M. on the same day it showed 41.25; or, in other words, the escaping gases were practically all CO. This state of things, however, did not last long, for the next analysis, at 5 o'clock that afternoon, gave a ratio of only 10, showing that reduction of the ore had begun.

This furnace at once began to do exceptional work; and it may be noticed here that on August 16th, just a week after the blast was first put on, the ratio fell below "the theoretical limit" of 2 and continued to be below that point, with a few exceptions, throughout the series of experiments. Indeed, for a period of two weeks, beginning September 2d, the ratio averaged only 1.50 and in one case fell to 1.35. During this time the furnace broke all previous records for low fuel, with one exception, and this with a production averaging nearly 20 tons of iron per 1000 cubic feet of capacity.

If now the diagrams be examined, with a view of determining what effect the different factors shown have on this ratio, one will be struck first by the degree to which it is influenced by the temperature of the escaping gases. In general, these two elements vary in direct proportion—indeed, the exceptions are so few that it might almost be said that a record of the temperatures of these gases would be equivalent to a record of analyses.

ILLINOIS STEEL COMPANY, Union Works, BLAST FURNACE No. 1.

August 9, to August 25,

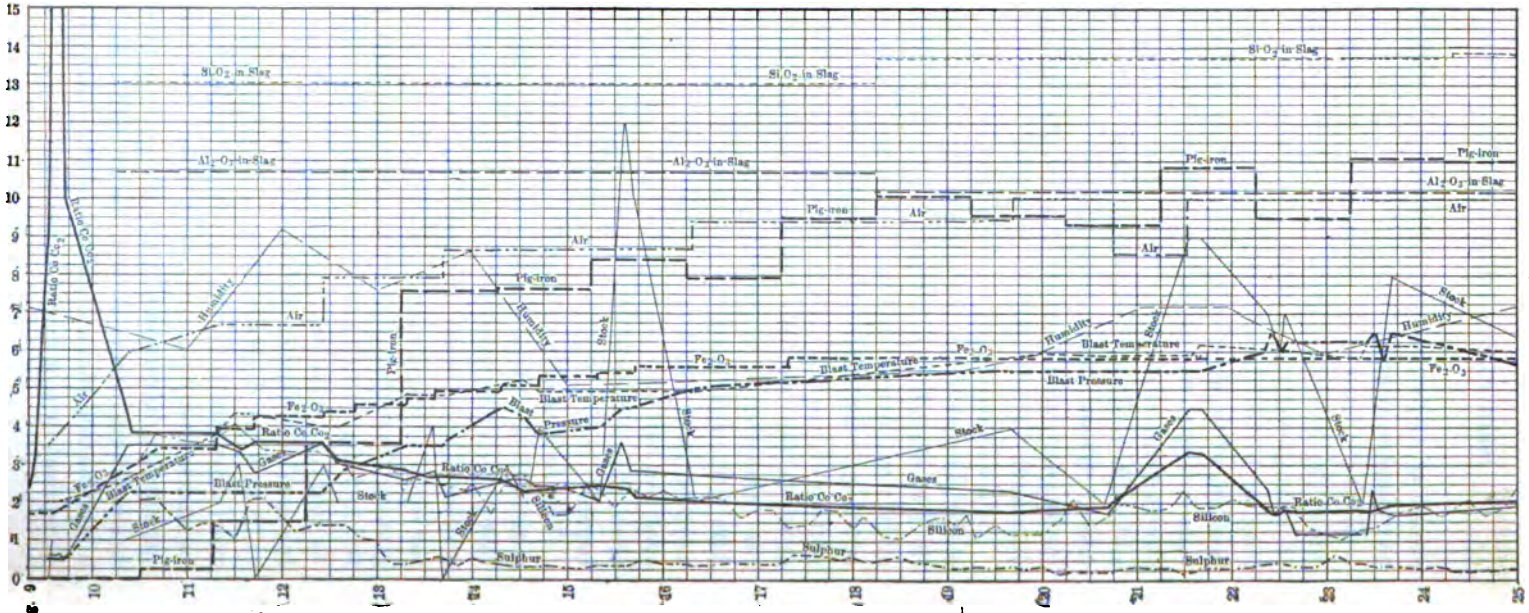
1890.

——— Ratio $\frac{CO}{CO_2}$ (by volume): Scale $\frac{1}{8}'' = 1$.
——— Temperature of escaping gases: Scale $\frac{1}{8}'' = 100°$ C.
– – – – Temperature of blast: Scale $\frac{1}{8}'' = 100°$ C.
– – – – Pressure of blast: Scale $\frac{1}{8}'' = 1$ lb.
——— Distance of stock from stock line: Scale $\frac{1}{8}'' = 1'$.
– – – – Relative humidity of atmosphere: Scale $\frac{1}{8}'' = 10\%$.
– – – – Weight Fe_2O_3 per 3600 lbs. **coke in charge: Scale $\frac{1}{8}'' = 1000$ lbs.**

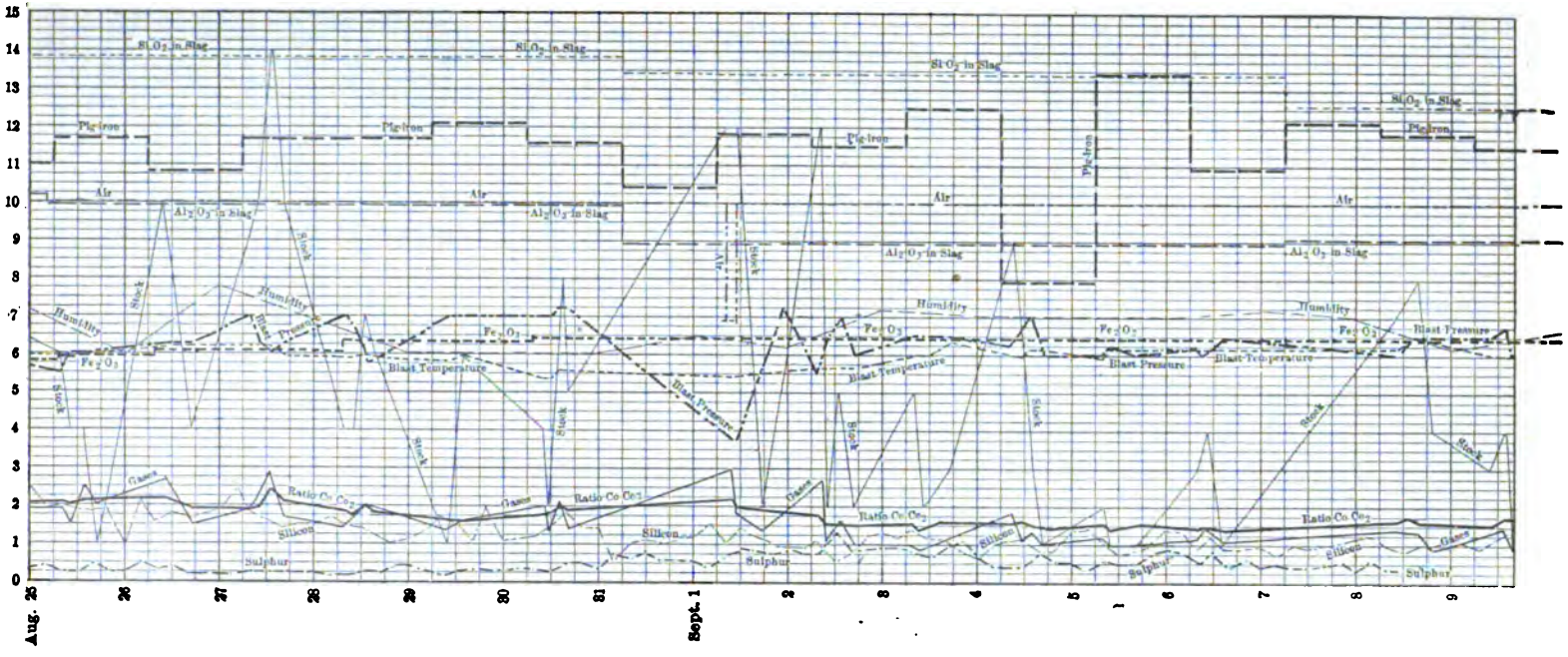
——— Yield of pig-iron per 24 hours: Scale $\frac{1}{8}'' = 10$ tons.
– · – · – Cubic feet of air per minute: Scale $\frac{1}{8}'' = 1000$ cu. feet.
– · – · – Per cent silicon in pig-iron: Scale $\frac{1}{8}'' = 1\%$.
– · – · – Per cent sulphur in pig-iron: Scale $\frac{1}{8}'' = .1\%$.
– – – – Per cent SiO_2 in slag: scale $\frac{1}{8}'' = 2\%$.
——— Per cent Al_2O_3 in slag: Scale $\frac{1}{8}'' = 2\%$.

NOTE:—No scrap used.

The main divisions on the horizontal line denote days, and the number at each is the day of the month beginning at midnight.



BLAST FURNACE No. 1. Continued.—August 25, to September 9, 1890.



ILLINOIS STEEL COMPANY, South Works, BLAST FURNACE No. 1.

October 27 to November 12,

1890.

Ratio $\frac{CO}{CO_2}$ (by volume): Scale $\frac{1}{8}'' = 1$.

Temperature of escaping gases: Scale $\frac{1}{8}'' = 100°$ C.

Temperature of blast: Scale $\frac{1}{8}'' = 100°$ C.

Pressure of blast: Scale $\frac{1}{8}'' = 1$ lb.

Distance of stock from stock line: Scale $\frac{1}{8}'' = 1'$.

Relative humidity of atmosphere: Scale $\frac{1}{8}'' = 10\%$.

Weight Fe_2O_3 per 6400 lbs. coke in charge: Scale $\frac{1}{8}'' = 1000$ lbs.

Yield of pig-iron per 24 hours: Scale $\frac{1}{8}'' = 20$ tons.

Cubic feet of air per minute: Scale $\frac{1}{8}'' = 1000$ cu. feet.

Per cent silicon in pig-iron: Scale $\frac{1}{8}'' = 1\%$.

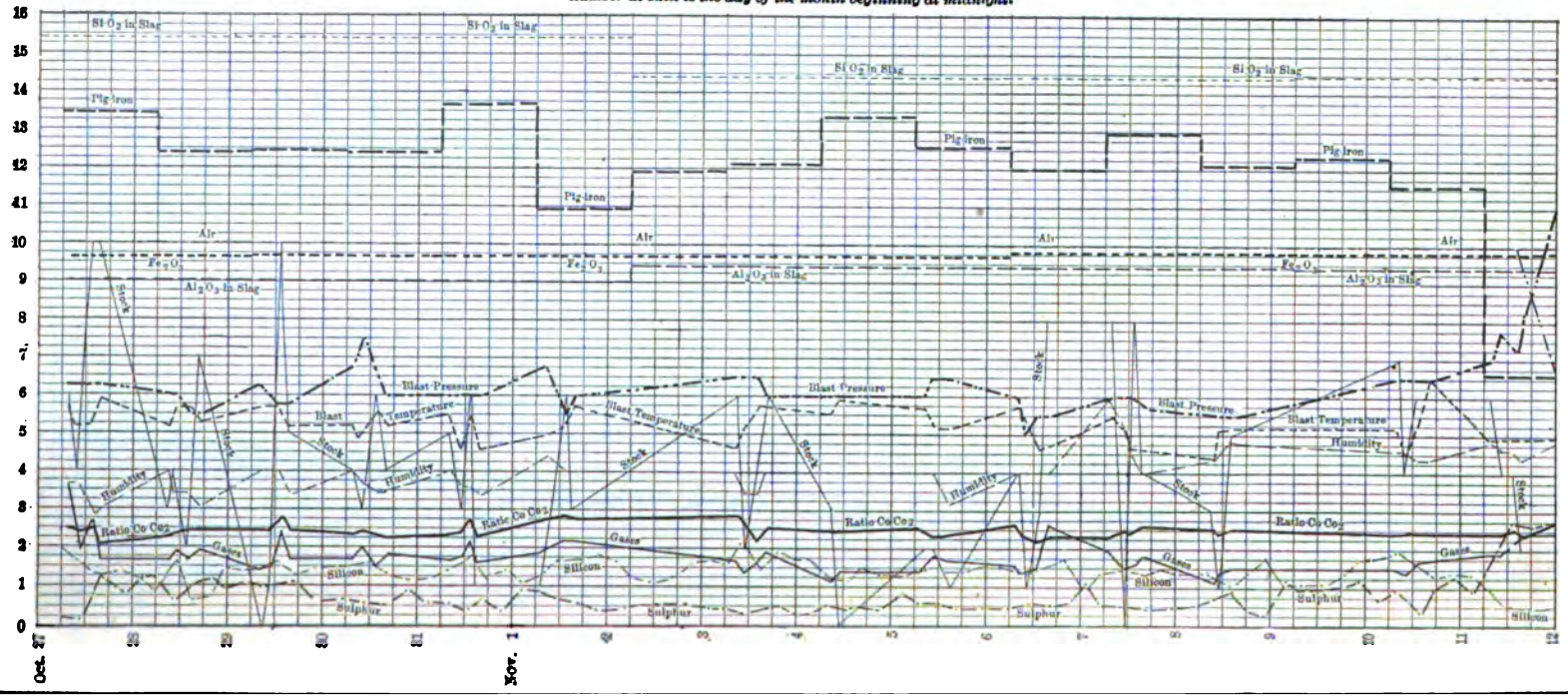
Per cent sulphur in pig-iron: Scale $\frac{1}{8}'' = .1\%$.

Per cent SiO_2 in slag: scale $\frac{1}{8}'' = 2\%$.

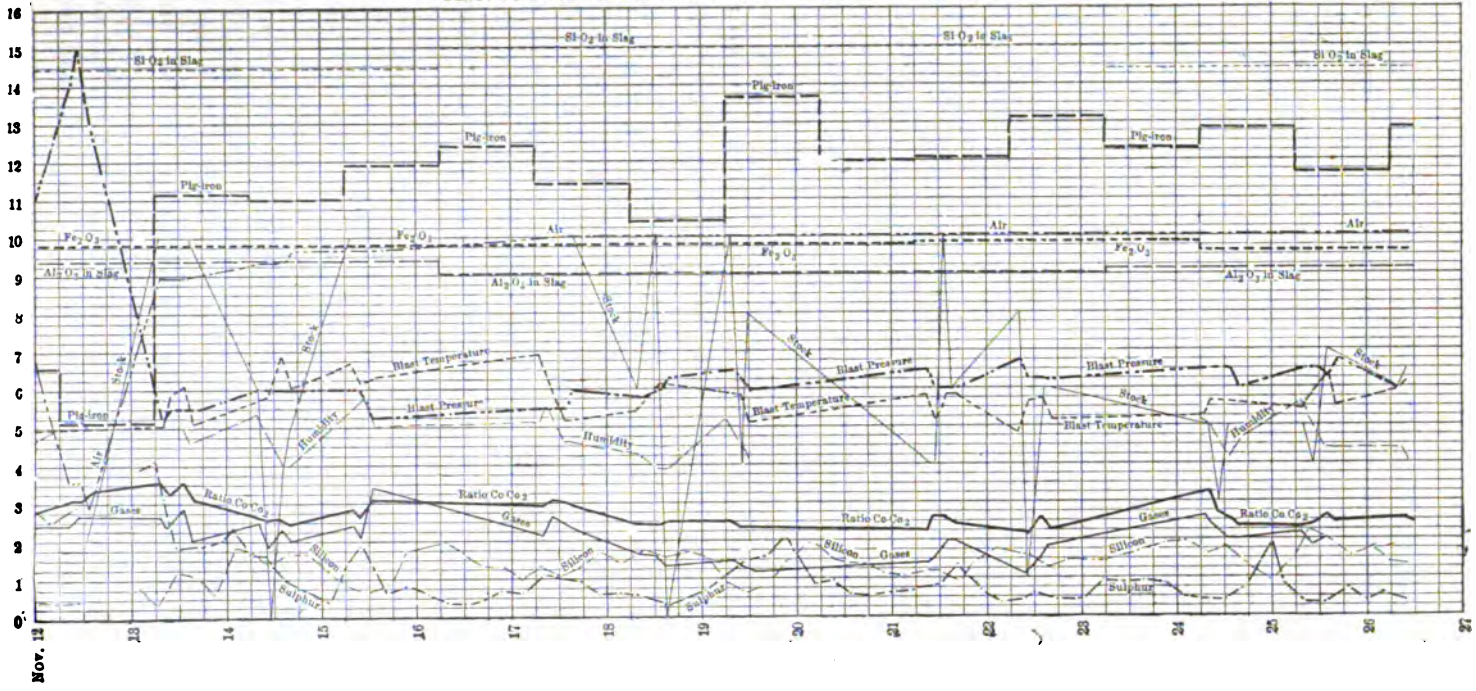
Per cent Al_2O_3 in slag: Scale $\frac{1}{8}'' = 2\%$.

NOTE:—1300 lbs. Scrap charged per 6400 Coke.

The main divisions on the horizontal line denote days, and the number at each is the day of the month beginning at midnight.



BLAST FURNACE No. 1. Continued.—November 12, to November 27, 1890.



A study of the line representing the relative humidity of the atmosphere shows that it follows, almost without exception, the main changes in the gas-ratio-lines. This is more clearly indicated in the two South Works furnaces, where the observations were made more frequently than at the Union furnace. It proves conclusively that any application of water-gas through the tuyeres cannot but have a harmful effect on the working of the furnace.

With regard to the other lines, it is hard to deduce any law, there being so many variables present that changes in the composition of the gases cannot be attributed to any one of the factors alone.

If now the ratio-lines on all the furnaces be compared, it will be seen that Union Furnace No. 1 (when working on a full burden, beginning, say September 2d) has by far the lowest and most regular line of all. That of South No. 2 comes next in height, with less regularity, while South No. 1 shows the highest and most irregular line. On referring to the following page, showing "Distribution of Heat," it will be seen that a great difference exists, both in shape and size, between the Union furnace and the two South Chicago furnaces, which are quite similar to each other. The former, while about equal in height, has a narrow bosh, a steep angle to the boshes, and, above all, a narrow stock-line and a correspondingly narrow bell. The latter two present quite opposite conditions.

Further down on the page it will be seen that there is a difference of 44 per cent. in the amount of carbon burned in the zone of reduction; a difference of 100° C. in the temperature of the escaping gases; and (most important of all) a difference of 13 per cent. in the total heat developed in the furnace per unit of carbon burned. Moreover, owing to the very regular composition of the gases at "Union," it follows that the composition of the iron would be correspondingly regular, making it much easier for the manager to run to a desired composition. These differences seem to be due chiefly to the width of the stock line—the wide-top at the South Works furnaces giving the gas a ready chance to "channel." This is significantly shown on the diagrams by the line representing pressure of blast, which (other things being equal and except when the furnace was hanging) varies inversely as the ratio-line. On the other hand, the narrow top of the Union furnace compresses the volume of the gas to such an extent that although it travels faster through the ore, yet it is forced through the entire mass of it. Reduction of the ore, therefore, takes place in a cooler part of the furnace, and for

that reason to a greater extent by CO. The cost of labor per ton of iron being the same in both places (it is, however, slightly in favor of Union), it follows that the Union style of furnace is much more economical than that of the South works.

Distribution of Heat.

Elements of the Furnace.	No. 1. Union.	No. 2. South.
Internal capacity of furnace,	6,700 cub. ft.	13,900 cub. ft.
Height,	72 ft. 4 in.	75 ft.
Bosh,	14 ft. 10 in.	20 ft.
Hearth,	8 ft. 6 in.	11 ft.
Stockline,	9 ft. 4 in.	15 ft. 3 in.
Yield in 24 hours,	130 tons.	223 tons.
Capacity per ton of iron per 24 hours,	51.5 cub. ft.	62 3 cub. ft.
Ore consumed per lb. of iron made,	1.57 lb.	1.64 lb.
Flux " " "	0.27 "	0.26 "
Slag produced " "	0.30 "	0.30 "
Total carbon burned per lb. of iron made,	.640 "	.745 "
Carbon burned in zone of reduction,	.066 "	.095 "
Average temperature of blast, degrees cent.,	650	570
Average temperature of escaping gases,	125	225
Ratio $\frac{CO}{CO_2}$ by volume,	1.50	2.38
Caloric developed per lb. of carbon burned,	4541 Cal.	4013 Cal.
Weight of blast,	3.132 lb.	3.551 lb.
Weight of gases,	4.451 "	5.101 "
Caloric of combustion in zone of reduction,	1487 Cal.	1384 Cal.
Caloric of " " tuyeres,	1420 "	1607 "
Total caloric of combustion,	2907 "	2991 "
Caloric carried in by blast,	478 "	484 "
Total caloric received,	3385 "	3475 "
Caloric of zone of fusion,	1898 "	2091 "
Caloric absorbed by reduction of ores and fusion of iron,	2314 "	2314 "
Caloric absorbed by fusion of slag,	*165 "	*165 cal.
Caloric absorbed by decomposition of stone, etc.,	241 "	265 "
Sensible heat carried off by gases,	132 "	272 "
Caloric lost by radiation, etc. [by diff.],	533 "	459 "
Total caloric consumed,	3385 "	3475 "

If now the lines of South Furnace No. 1 be compared with those of South No. 2, it will be seen, first, that the ratio-line is much higher and more irregular in the former than in the latter. Moreover, the line showing the temperature of the escaping gases is much

* This figure is probably too low, as the figures taken [550] for the specific heat of slag applies to a much more fusible slag than the one made at these furnaces.

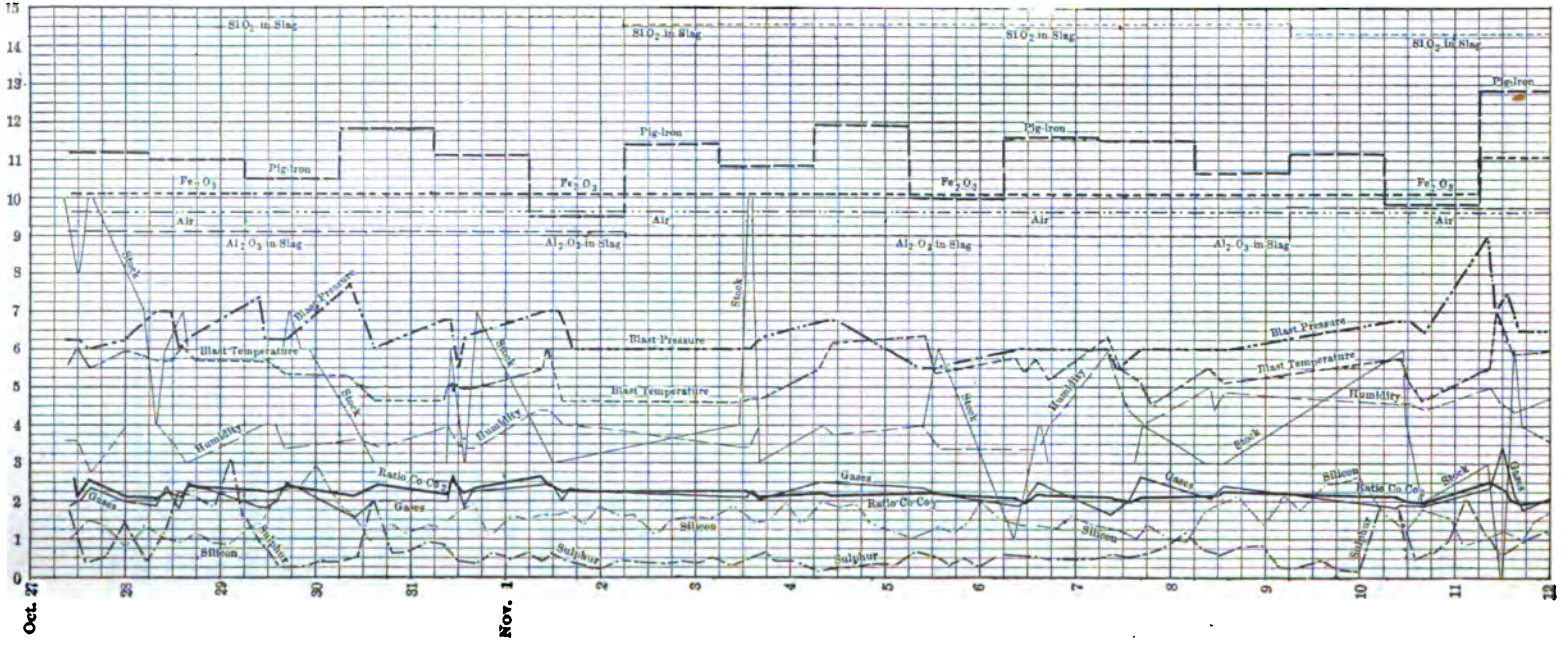
ILLINOIS STEEL COMPANY, South Works, BLAST FURNACE No. 2.

October 27 to November 12, 1890.

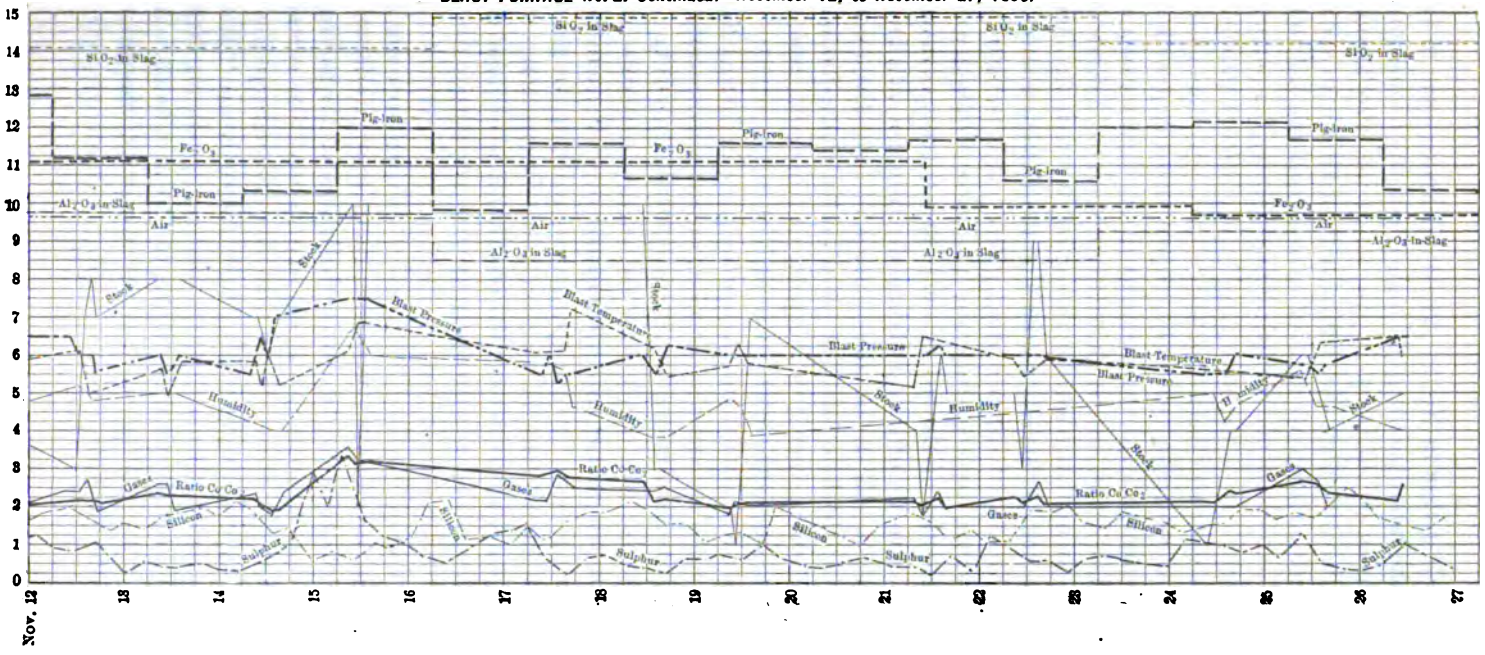
- Ratio $\frac{CO}{CO_2}$ (by volume): Scale ½″ = 1.
- Temperature of escaping gases: Scale ½″ = 100° C.
- Temperature of blast: Scale ½″ = 100° C.
- Pressure of blast: Scale ½″ = 1 lb.
- Distance of stock from stock line: Scale ½″ = 1′.
- Relative humidity of atmosphere: Scale ½″ = 10%.
- Weight Fe_2O_3 per 6400 lbs. coke in charge: Scale ½″ = 1000 lbs.
- Yield of pig-iron per 24 hours: Scale ½″ = 20 tons.
- Cubic feet of air per minute: Scale ½″ = 1000 cu. feet.
- Per cent silicon in pig-iron: Scale ½″ = 1%.
- Per cent sulphur in pig-iron: Scale ½″ = .1%.
- Per cent SiO_2 in slag: Scale ½″ = 2%.
- Per cent Al_2O_3 in slag: Scale ½″ = 2%.

NOTE:—No scrap used.

The main divisions on the horizontal line denote days, and the number at each is the day of the month beginning at midnight.



BLAST FURNACE No. 2. Continued. November 12, to November 27, 1890.



lower in No. 1 than in No. 2. These differences seem to be almost constant, the ratio-line being .44 higher, and the temperature of the escaping gases 50° C. lower in No. 1. It was also found that in changing the burden on No. 1 furnace from all ore to one carrying 1300 pounds of scrap per 6400 of coke, enough ore had to be taken off to leave an excess of 1 pound of coke to every 3.2 pounds of iron in the charge. In other words, while in ordinary cupola-practice 1 pound of coke will melt 10 pounds of iron, yet in the blast-furnace 1 pound of coke will melt only 3.2 pounds of iron, or it requires three times as much coke to melt iron in the blast-furnace as in the cupola. This seems, at first glance, surprising, but I think it can be explained, in part at least, by supposing the metallic iron in the blast-furnace to be partially oxidized, in the upper region of the furnace, by the CO_2 of the ascending gases. This being a heat-producing reaction, the temperature of this zone is raised, and consequently reduction of the ore by solid carbon becomes more active. The Fe_xO_y thus formed requires heat for reduction at a lower point in the furnace. Of course, the quantity of heat given out by the oxidation of the iron and that absorbed by the reduction of the Fe_xO_y is the same, but it is the point at which the reactions take place which causes the total loss in fuel.

The lesser temperature of the escaping gases in furnace No. 1 may be explained by the fact that metallic iron has not the power, at least to so great an extent as ore, to split up 2CO into $C + CO_2$. Besides, this reaction depends largely upon the length of that zone which lies between the temperatures of 450° F. and 800° F.; and as this is shorter, owing to the reasons given above, in the furnace working on scrap, it follows that the reaction would be less intense in that furnace. As this is an economical reaction, it is obvious that here is another reason for the excessive fuel-consumption in the furnace running on scrap.

Union Works Furnace No. 1, after having been on iron six months, was put on 83 per cent. ferro-manganese. It did as well on this as it had done on iron. Several analyses of the gases gave the ratio of CO_2 to CO at 1 to 4.2, and showed that they contained, on an average, 5.36 per cent. of the total manganese charged, as against 10 per cent., mentioned by Mr. H. M. Howe in his "Metallurgy of Steel." Later, when the same furnace was producing 30 per cent. spiegel, the gases gave a ratio of CO_2 to CO at 1 to 2.3. Analyses then showed that 1.68 per cent. of the manganese charged was carried off in the gases.

MANGANESE IN CAST-IRON.

BY W. J. KEEP, DETROIT, MICH.

(Cleveland Meeting, June, 1891.)

MANGANESE is a nearly white metal, having about the same appearance when fractured as white cast-iron. Its specific gravity is about 8, while that of white cast-iron, reasonably free from impurities, is but a little above 7.5. As produced commercially, it is combined with iron, and with small percentages of silicon, phosphorus, and sulphur.

It is generally produced in the blast-furnace. If the silicon is under 0.50 per cent., the product will be white. If the manganese is under 40 per cent., with the remainder mostly iron, and silicon not high enough to make the product gray, the alloy is called spiegeleisen, and the fracture will show flat reflecting surfaces, from which it takes its name.

With manganese above 50 per cent., the iron alloy is called ferro-manganese.

As manganese increases beyond 50 per cent., the mass cracks in cooling, and when it approaches 98 per cent. the mass crumbles or falls in small pieces.

When the percentage of manganese is high, the planes of the fracture are not so bright as when it is lower.

The ferro-manganese which we have used in the experiments reported in this paper, contains Mn, 81.62; Si, 0.256; and combined carbon, 6.153 per cent. We have not been able to make test-bars from this, because the alloy cracks in small pieces in cooling.

Manganese combines with iron in almost any proportion, but if an iron containing manganese is remelted, more or less of the manganese will escape by volatilization, and by oxidation with other elements present in the iron. If sulphur be present, some of the manganese will be likely to unite with it and escape, thus reducing the amount of both elements in the casting. Hence, in remelting irons, some of the sulphur which has entered from the fuel with which the iron is melted will very likely unite with some of the manganese in irons which make up the mixture and escape. Remelting, therefore, reduces the percentage of manganese in cast-iron.

The tests here recorded have been made by taking various grades of cast-iron and adding ferro-manganese, so as to produce, as nearly as possible, a given percentage of manganese in the final casting to be examined. Owing to the escape of manganese above referred to by volatilization and oxidation, the actual amount of manganese is, perhaps, somewhat below the percentages given by us. The percentages given in the tables are what we calculated would be in the casting if all remained.

The methods of testing employed in these experiments are described in a previous paper (*Trans.*, xviii., 798).

Cast-iron, when free from manganese, cannot hold more than 4.50 per cent. of carbon, and 3.50 is as much as is generally present; but as manganese increases, carbon also increases, until we often find it in spiegel as high as 5 per cent., and in ferro-manganese as high as 6. This effect on capacity to hold carbon is peculiar to manganese.

Because the spiegeleisen and ferro-manganese of commerce are white, it seems to have been taken for granted that the combined state of carbon was caused by the manganese present, and because combined carbon may be called dissolved carbon, it has often been declared that manganese possesses the power of dissolving carbon.

That this view is erroneous, we shall fully prove, showing that the condition of the carbon in combinations of manganese and iron is determined by the percentage of silicon present, just as in any other cast-iron.

To be able to investigate the influence of manganese in cast-iron, we must be familiar with the action of silicon. This is described at length in former papers (*Trans.*, xvii., 253 and 683). It is sufficient to say here that castings of iron alone, or iron and carbon, will always be white, and the carbon will always be combined. If the iron be saturated with carbon, and comparatively free from other elements, such cast-iron will be crystalline, and the fracture will be more or less specular.

The presence of silicon will drive more or less of the carbon into the graphitic form, and cause the color of the cast-iron to be gray. Practically speaking, we may say that the grayness of cast-iron depends on the percentage of silicon present.

If we introduce into the iron an element which will drive out or lessen the percentage of silicon, such element will indirectly cause the cast-iron to be whiter; but such whiteness should be ascribed to the decrease of silicon. However, it may be that manganese, either

by allowing an increased amount of combined carbon to be present, or by making the combined carbon less easily influenced by silicon, does, in a slight degree, cause iron containing a given percentage of silicon to be less gray than if manganese were absent, or the manganese may, by its own white color, lighten the shade of cast-iron.

I. After making numerous tests, presently to be described, we made the following three casts to determine the exact influence exerted by silicon, and to prove the correctness of the opinion just expressed that the change in silicon produced much of the change observed in our other records.

We prepared three hot ladles, and placed in the first, five ounces of broken gray scrap from our ordinary cupola-iron.

In the second ladle we placed four ounces of white iron, containing practically no silicon, and one ounce of 14.77 per cent. ferro-silicon.

In the third ladle we placed four ounces of 81.62 per cent. ferro-manganese (white) and one ounce of the ferro-silicon.

Into each ladle we tapped from the cupola as nearly as possible the same amount of melted iron. As the cupola-iron contained 2.50 per cent. silicon, the ounce of ferro-silicon kept the percentage of silicon practically uniform in all the casts, since it supplied enough silicon to make an average of 2.51 with the four ounces of other material added in the second and third. In the third cast the manganese was increased 1.08 per cent.; and as manganese in the cupola-iron was about 0.50, in the third cast it was about 1.50.

All conditions attending each cast were exactly the same, and, as will afterwards be seen, our other tests and this series substantially agree.

A comparison of the first and second casts will show whether any influence was exerted by the introduction of the white iron and ferro-silicon.

A comparison of the second and third casts will show what influence the manganese exerted.

The record of stress applied as a dead load (or, as we call it, "weight-stress") shows that the iron is neither weakened nor strengthened by manganese; but the record of impact-stress shows that the addition of 1 per cent. of manganese has lessened the ability of a casting to resist shock.

A few general remarks may be made regarding deflection and set, as measured in these tests. Deflection is the sum of set and elas-

ticity, set being the portion of deflection which will not be recovered; and elasticity the portion which will be recovered when the stress is removed. In most cases deflection for impact exceeds that for weight; and also, in most cases, set is less for impact than for weight—the reason being that the suddenness of a blow does not allow time for the separation of the molecules before the recoil takes place. If the ultimate stress is less in any test we might expect a corresponding reduction in deflection and set. For purposes of comparison we must use the ratio obtained by dividing set and deflection by the stress which produced them.

TABLE I.—*Results of Series I.*

Number of cast.	Mangan'se added.	Silicon added.	Weight-stress.	Weight-deflect'n.	Weight-set.	Impact-stress.	Impact-deflect'n.	Impact-set.	Shrinkage.	Chill.	Hardness.	Hardness of chill.
1	0.	2.50	380	.190	.029	365	.225	.022	.121	.075	70	124
2	0.	2.51	398	.195	.029	323	.225	.030	.123	.060	70	124
3	1.08	2.51	380	.165	.017	271	.182	.015	.155	.065	99	160

The ratios of sets show that manganese renders cast-iron less plastic and more brittle. We see by the record of impacts that the lessened ability of cast-iron to resist shock is an indication of brittleness. By subtracting the ratio of set from that of deflection we have the ratio of elasticity, and we find that elasticity is not influenced materially by manganese.

The next question is that of shrinkage. All of our tests agree in showing that manganese increases the shrinkage of cast-iron. In the tests which we are considering, an increase of 1 per cent. has raised the shrinkage 26 per cent. So long as any carbon remains in the combined form, an increase in silicon will drive such carbon into graphite, and thus in some degree decrease shrinkage; but a high shrinkage caused by manganese is independent of carbon, and cannot be taken out without removing the manganese. As shrinkage varies with the size of the casting, and produces internal stress within the casting, this question is of vital importance to the foundryman. The less shrinkage in the iron, the less the danger from cracks.

Perhaps the next important question for the foundryman is that of chill. Judging from these records, manganese does not influence

chill at all; but other tests will show, as we have before observed, that with a given percentage of silicon the carbon may be a little more inclined to remain in the combined form, and therefore the chill may be a little deeper. Hence, to cause the chill to be the same, it would seem that the percentage of silicon should be a little higher with manganese than without it.

Hardness is another important question with the worker in iron. An increase of 1 per cent. of manganese has increased the hardness 40 per cent. In the first two casts the hardness of the chill is 77 per cent. greater than that of the unchilled fracture; but in the cast with manganese the difference is only 61 per cent.—again showing that manganese does not increase chill. If, however, a hard chill is required, manganese gives it by adding hardness to the whole casting. This hardness is due to the hardness of the manganese itself, and not because more of the carbon has taken the combined form.

This preliminary review presents a key by which to examine the records of other series of tests which are presented below.

It seems that in trying to make soft castings with low shrinkage, manganese should be avoided.

II. To prove still further the influence of silicon in manganese-irons, and that much of the influence exerted by manganese can in most cases be overcome by silicon, we present, in Table II., the record of a series of six tests with 20 per cent. spiegeleisen.

TABLE II.—*Results of Series II.*

Percentages Manganese.	Silicon added.	Weight-stress.	Weight-deflect'n.	Weight-set.	Impact-stress.	Impact-deflect'n.	Impact-set.	Shrinkage.	Chill.	Hardness.	Hardness of chill.
20.00	0.	150	.05	.01	68	.09	.01	.272	White.	185	235
20.00	1.66	168	.07	.02	101	.11	.02	.266	0.	186	235
20.00	3.00	179	.16	.05	101	.14	.03	.240	0.	146	215
20.00	4.84	230	.27	.11	118	.21	.06	.222	0.	115	160
20.00	6.20	326	.47	.24	350	.44	.19	.218	.001	123	173
20.00	7.30	195	.30	.12	169	.24	.07	.213	.075	127	265

The first cast is from the spiegel alone, and contains hardly any silicon. The fracture is white with the usual reflecting surfaces.

The second contains 1.66 per cent. of silicon, which has changed the metal to a light gray, with no tendency to chill, and has almost wholly destroyed the specular fracture presented by the spiegel.

The third contains 3.00 per cent. of silicon, which turns the metal into a dark gray open-iron with no chill.

The fourth contains 4.84 per cent. of silicon, which has caused the grain to be slightly closer and a little lighter in color, but with absolutely no chill.

The fifth contains 6.20 per cent. of silicon, which has caused the grain to be still finer and lighter. The casting was perfectly sound. There was no chill, but there was a white skin on the chill-end, which could scarcely be seen.

The sixth contains 7.30 per cent. of silicon. This still further lightens the color of the fracture, and shows the bright facets of a silvery iron. The grain of this casting is very open, and the fracture is full of cavities filled with flakes of graphite. These bars have a chill 0.05 to 0.075 inch deep. We have formerly called this chill in high-silicon irons, a "silicon chill."

It seems as if the presence of this 20 per cent. of manganese prevented the combination of silicon and carbon from forming a chill, with low percentages of silicon. This series gives an intimation that, with high manganese, perhaps similar combinations between silicon and carbon may form, but not until a large percentage of silicon is reached; or, in other words, manganese being present, a larger percentage of silicon is required to produce a given result.

The hardness of this series is remarkable, proving again the hardening properties of manganese. The castings with 7.30 silicon look like the softest of open gray-iron, but can hardly be touched with a file.

We have shown, in a former paper, that silicon of itself hardens iron. We see, in this case, that after the softening due to graphite has produced its greatest effect, the hardening influence of silicon was felt in the last two casts.

We shall have more to say regarding this series.

We will now describe several series of tests which we have made during the last four years. The records are given below, both in tabular form and graphically.

III. We took (as a base) a Swedish gray charcoal pig-iron "$F^{L}M$" containing 1.249 silicon and 0.187 manganese. We made one set of test-bars from this. We then took another quantity of this $F^{L}M$ iron and added enough 81.62 per cent. ferro-manganese to make the

addition of manganese 0.12 per cent. We made other casts (see Table III.), with 0.66, 0.75, and 1.00 per cent. of manganese added.

TABLE III.

Mangan'se added.	Silicon estimat'd.	Weight-stress.	Weight-deflect'n.	Impact-stress.	Shrinkage.	Chill.
0.12	1.247	343	.23	282	.159	.40
0.66	1.239	297	.18	226	.161	.40
0.75	1.237	342	.16	259	.164	.50
1.00	1.233	334	.15	237	.168	.70

F[L]M being a charcoal-iron and low in silicon, is very sensitive to any element which can exert an influence on carbon.

The additions of ferro-manganese diminished the silicon in each cast. The effect of this reduction must be considered when the record of this series of tests is examined.

When the series is considered as a whole, "weight-stress" is neither increased or diminished.

As manganese was increased, the bars had less ability to withstand shock; deflection was decreased materially, and shrinkage increased uniformly with each increase of manganese. The chill increased, but not more than would be expected from the reduced amount of silicon.

TABLE IV.—*Results of Series IV.*

Mangan'se added.	Silicon estimat'd.	Weight-stress.	Weight-deflect'n.	Weight-set.	Impact-stress.	Impact-deflect'n.	Impact-set.	Shrinkage.	Chill.	Hardness.
0.50	1.53	353	.27	.08	373	.29	07	.152	.02	62
1.66	1.51	357	.28	.06	381	.26	.05	.156	.05	99
2.26	1.50	329	.21	.03	305	.22	.03	.157	.20	99
3.00	1.48	380	.28	.06	313	.24	.04	.142	.25	...
3.83	1.47	331	.26	.05	330	.25	.04	.136	.20	...
10.88	1.36	215	.09	.01	118	.11	.01	.232	.50	...

IV. We next took as a base "Stewart" coke-iron, which we

thought would not be so sensitive as F^{L}M. The pig used was said to contain 1.53 silicon and about 0.50 manganese. The results shown in Table IV. were not at all satisfactory, as we do not find a uniform variation in the series. We added the ferro-manganese as near the time of pouring as possible, to prevent the escape of the manganese.

As in case III., the silicon was diluted with each addition of manganese, and such dilution will account for some of the variations.

TABLE V.—*Results of Series V. and VI.*

Mangan'se added.	Silicon estimat'd.	Weight-stress.	Weight-deflect'n.	Weight-set.	Impact-stress.	Impact-deflect'n.	Impact-set.	Shrinkage.	Chill.	Hardness.	Hardness of chill.
SERIES V.											
0.	2.50	374	.19	.03	222	.18	.01	.143	.03	...	...
0.75	2.48	384	.17	.02	339	.21	.02	.144	.05	...	...
1.62	2.45	380	.17	.01	204	.15	.01	.154	.07	...	...
3.00	2.41	Discount			254	.17	.01	.159	.15	...	...
(7.66)	(2.28)	380	.20	.03	220	.17	.01	.162	.25	...	...
(11.50)	...	322	.17	.03	288	.20	.02	.168	.25	...	...
(13 25)	...	400	.27	.08	237	.19	.03	.191	.25	...	...
SERIES VI.											
0.	2.50	364	.18	.01	271	.19	.02	.147	.10	90	195
0.75	2.48	420	.19	.02	254	.18	.01	.148	.18	103	187
1.87	2.44	394	.16	.01	220	.17	.01	.159	.15	111	235
3.50	2.39	323	.14	.01	204	.15	.01	.165	.12	124	215
(9.00)	(2.24)	378	.17	.01	237	.17	.01	.163	.20	116	219
(11.00)	...	354	.16	.03	237	.20	.04	.185	.50	120	290
(20.00)	...	289	.16	.03	204	.17	.02	.177	.40	137	265

MEM.—In the last three casts of Series V. and VI. the ferro-manganese was not all melted, and we cannot, therefore, compute the percentages of manganese and silicon.

The variableness makes it impossible to draw conclusions from this series alone, but it does not in any way cause us to alter our

conclusions already reached. Shrinkage ran up until 2 per cent. of manganese was reached, but decreased with 3 and 3.83 per cent. With 10 per cent. manganese, the silicon was diluted to 1.36, and the casting was porous and light colored. Chill was difficult to determine, but was about 0.50.

V. and VI. These, shown in Table V., were made by placing in a moulder's ladle the amount of ferro-manganese (heated red) necessary to produce the desired percentage of manganese in the casting, and tapping molten iron from the cupola, containing 2.50 per cent. silicon. The melted iron was poured immediately into the moulds. The time of making all the seven casts was not more than five minutes, therefore the manganese could not have had time to escape, and aside from Mn (and diluted Si) the composition of each cast was the same. The two sets of tests were made some three months apart. The additions in these casts also diluted the silicon as manganese increased.

These two series sustain our conclusions in every particular.

TABLE VI.—*Results of Series VII. and VIII.*

SERIES VII.

Manganese added.	Weight-stress.	Weight-deflection.	Shrinkage.
0.	433	.21	.127
0.25	395	.17	.137
0.50	402	.18	.141
0.75	379	.15	.154
1.00	422	.18	.158
1.25	387	.16	.155
1.50	402	.17	.163
1.75	440	.19	.162
2.00	435	.17	.159

SERIES VIII.

Manganese added.	Weight-stress.	Weight-deflection.	Shrinkage.
0.	440	.25	.130
0.25	430	.21	.141
0.50	405	.21	.150
0.75	393	.20	.147
1.00	430	.22	.159
1.25	392	.19	.160
1.50	409	.18	.160
1.75	401	.19	.168
2.00	433	.22	.161

VII. and VIII. These (Table VI.) are similar to the last, and made in the same way, only that one cast was made each day to get a given percentage of manganese. At the same time another cast was made with the cupola-iron alone; so that each day we had two

casts exactly similar, except that one contained the manganese addition. In every case the manganese caused an increase in shrinkage. There was hardly any variation in chill, and what there was could be accounted for by dilution of silicon. The record of weight-stress was very irregular. That of impact we have not recorded. Shrinkage and deflection confirm former conclusions.

IX. This series of tests (Table VII.) was made in a different manner; namely, by placing 30 pounds of $F^{L}M$ in a crucible, melting it, making a cast of test-bars, adding some more $F^{L}M$, returning the gates from last cast, and adding enough ferro-manganese to make the percentage of manganese 0.25; melting this, and casting a set of bars; adding more $F^{L}M$, returning sprues, adding ferro to make manganese 0.50; making another set of bars; and so on for each $\frac{1}{4}$ per cent. up to 2 per cent. manganese. To produce the nine casts the metal was exposed to the heat of the furnace about nine hours.

The silicon was diluted by each addition, and no doubt some escaped. Analyses showed that in the last cast, where we meant to have 2 per cent. of manganese, there was only 0.5836 per cent., and of this the $F^{L}M$ in the pig contained 0.186.

X. We made a set of bars in the crucible from $F^{L}M$, with enough ferro-manganese to make manganese 2 per cent., adding the ferro just before pouring, to prevent escape. The silicon was diluted to 1.22 per cent.

This gave (Table VII.) exactly what was expected of the 2 per cent. cast of series IX. Comparing this set of bars (X.) with the first set of bars in series IX., or with series XI., we have $F^{L}M$ with 0.187 per cent. manganese and with 2 per cent., and silicon lowered by dilution only from 1.249 to 1.22. The records of these two casts are similar to set I. of these casts, only there is always a chance of considerable variation in separate casts made in a crucible, owing to unseen variations in the conditions attending the operation.

The similarity between these two heats proves that the reason why the last cast of series IX. was not like X. was the decrease of silicon and other changes produced by the continuation of the heat through the period of nine hours. The manganese has not exerted any influence, which is apparent when we compare the first cast of IX. or series X. with XI.

XI. We made (Table VII.) a series exactly similar to IX., without adding manganese (and without thus reducing the amount of silicon). The silicon was oxidized to such an extent that the ninth cast was white, with a corresponding variation in each of the casts.

Table VII. gives the silicon in each of the series. In IX. it is calculated, while in XI. it is determined by analysis of the bars.

TABLE VII.—*Results of Series IX., X. and XI.*

SERIES IX*.

Mangan'se added.	Manganese by Analyses.	Silicon estimated.	Silicon by Analyses.	Weight-stress.	Weight-deflect'n.	Impact-stress.	Impact deflec'tn.	Shrinkage.	Chill.
0.	In Pig 0.187	1.25	...	337	.18	314	...	.172	0.50
0.25	...	1.25	....	360	.24	330	....	.164	0.52
0.50	....	1.24	...	368	.24	339	...	.164	0.65
0.75	...	1.24	...	398	.18	314	....	.167	0.80
1.00	....	1.24	....	398	.17	382	...	.178	1.00
1.25	....	1.23	...	441	.22	457	....	.187	1.25
1.50	....	1.22	...	325	.13	296	...	.195	1.50
1.75	...	1.00	....	431	.17	399	....	.205	White.
2 00	0.583	.89	....	284	.09	220	...	.230	White
TEST X.									
2.00	....	1.22	....	404	.18	357	.22	.171	.80
SERIES XI.—FLM RE-MELTS.†									
...	In Pig 0.187	In Pig 1.25	1.16	404	.16	220	.16	.176	1.00
...	...	...	...	400	.20	330	.22	.180	.80
...	...	....	...	405	.25	339	.27	.179	.80
...	...	....	...	399	.22	365	.27	.183	1.25
...	...	....	...	420	.22	331	.24	.186	1.75
...	...	...	...	479	.23	340	.22	.208	White
...	...	...	...	411	.17	423	.28	.187	1.75
...	...	...	0.97	397	.16	287	.20	.208	White

* The determinations in series IX. were made by Harry S. Fleming, who intended, when these experiments were begun, to analyze products, and to appear as joint-author of the paper.

† The determination iu series XI. were made by Prof. C. F. Mabery, of Cleveland, O.

Comparing the two series, we find that the corresponding casts of each series are substantially alike, showing, therefore, but little influence from the manganese.

We have made, at various times, various other tests, some of which we present in Table VIII.

XII. To F^{L}M we added (Table VIII.) a ferro-silicon containing 12.08 silicon and 0.75 manganese, to make silicon 2.50. The grain was very dark, but white patches occurred in the casting, with dark gray iron on the outside.*

XIII. This is another test exactly like XII.

XIV. We took F^{L}M and added a ferro-silicon ("Govan") said to contain 9.87 per cent. silicon and 2.20 manganese, to make silicon 2.50 and manganese 0.56. The grain was very uniform.

XV. We made a mixture exactly like XII. and XIII., except that we added enough ferro-manganese to make Mn the same as XIV. The result was a casting with perfectly even grain.

XVI. We took iron from our ordinary cupola-mixture, with silicon about 2.50 and manganese under 1 per cent., which gave "weight-stress" about 375. Enough 20 per cent. spiegel was added to make manganese 4 per cent. This was melted in a crucible and a set of test-bars was made.

By the addition, silicon was diluted to 2.10 per cent. The strength was slightly increased, which would have been expected from the lowering of the silicon. The chill was also increased from 0.075 to 0.50, which can be accounted for in the same way. Shrinkage increased from .130 to .175. The grain was gray and open.

XVII. In another test we took F^{L}M (Si, 1.249) and added enough 40 per cent. ferro-manganese to make manganese 16 per cent. and silicon 0.75. The F^{L}M was strong and gray. The mixture was white and full of blow holes, and had but little strength. The mere decrease of silicon would have turned the iron white, but not necessarily with blow-holes. The quality of the ferro-manganese exerts a great influence upon the resultant casting.

XVIII. We made a cast of 81.62 per cent. ferro-manganese, with enough 14.77 per cent. ferro-silicon to make Mn 51.70, silicon 4.83, carbon about 4.00 and iron about 39.47. The resultant casting was so weak as to be easily broken in the fingers, and was perfectly white and had a greasy feel.

This amount of silicon with the 20 per cent. spiegel produced a

* The ferro-silicons used in these experiments were analyzed by Edward Orton Jr.

TABLE VIII.—*Results of Series XII. to XVIII.*

Series Number.	Manganese added.	Silicon added.	Weight-stress.	Weight-deflection.	Weight-set.	Impact-stress.	Impact-deflection.	Impact-set.	Shrinkage.	Chill.	Hardness.	REMARKS.
XII.	0.25	2.50	341	.24	.05	314	.24	.05	.140	...	45	Silicon and Manganese added by 12.08 per cent. Ferro-Silicon (Mn 0.75).
XIII.	0.25	2.50	375	.20	...	322	.24	.05	.144	.35	60	Si and Mn added by 12.08 per cent FeSi.
XIV.	0.56	2.50	367	.28	.10	384	.30	.07	.147	.04	48	Silicon and Manganese added by 9.87 per cent. "Govan" Ferro-Silicon (Mn 2.20).
XV.	0.56	2.50	382	.23	.05	398	.26	.04	.161	.17	45	Silicon added by 12.08 per ct. FeSi. Manganese added by 81.62 per cent. FeMn.
XVI.	4.00	2.10	336	.17	.03	202	...	...	.173	.50	...	Manganese added by 20 per ct. Spiegeleisen.
XVII.	16.00	.75	Full of blow holes				...	...	.235	.75	...	Manganese added by 40 per cent. Ferro-Manganese.
XVIII.	51.70	4.83	Broke in cooling			...	...	...	.230	White	...	Manganese added by 81.62 FeMn. Silicon added by 14.77 FeSi.

MEM.—The cupola-iron from which Series XVI. was made had 2.50 Si with shrinkage .130, and chill .075.

beautiful open, gray casting with absolutely no chill. We did not,

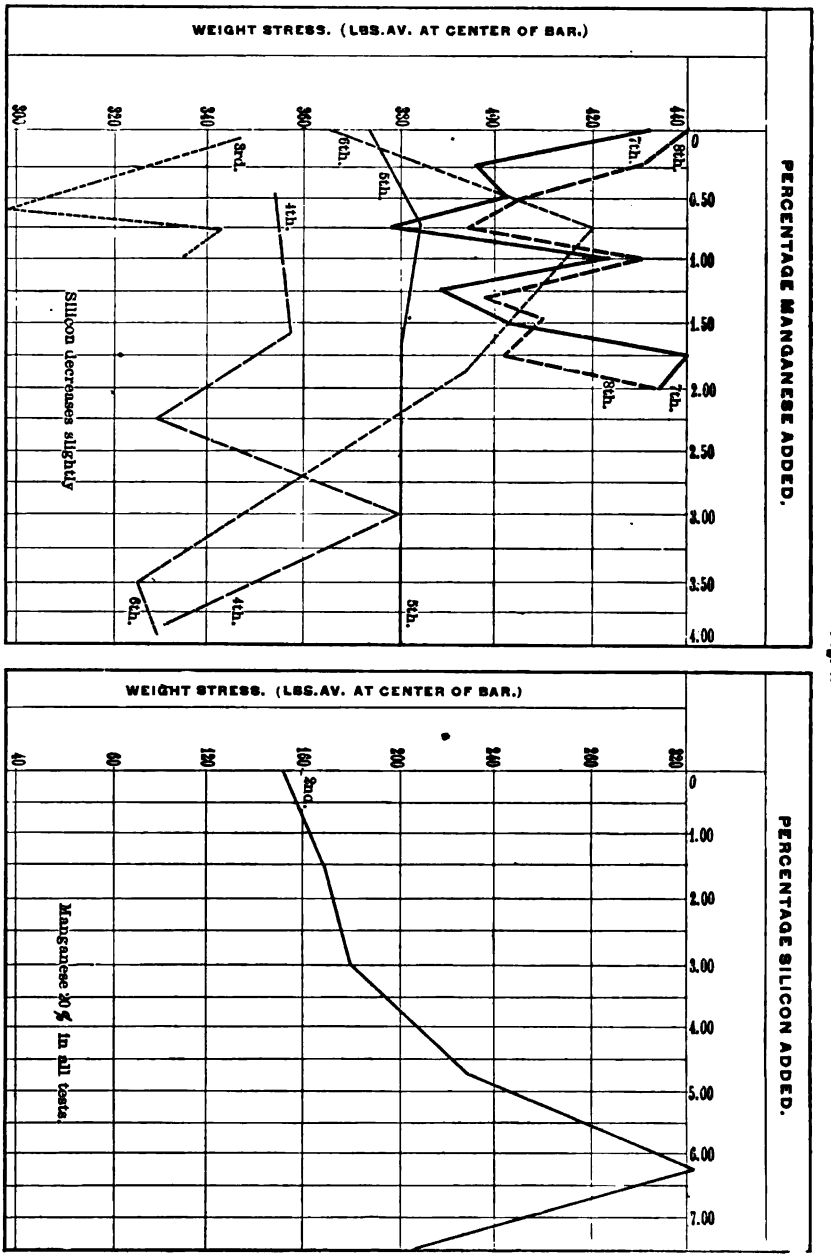


Fig. 1.

in XVIII., push the increase of silicon farther, and do not know if the casting with more silicon would have turned gray.

We have the record of tests of 27 different brands of cast-iron of

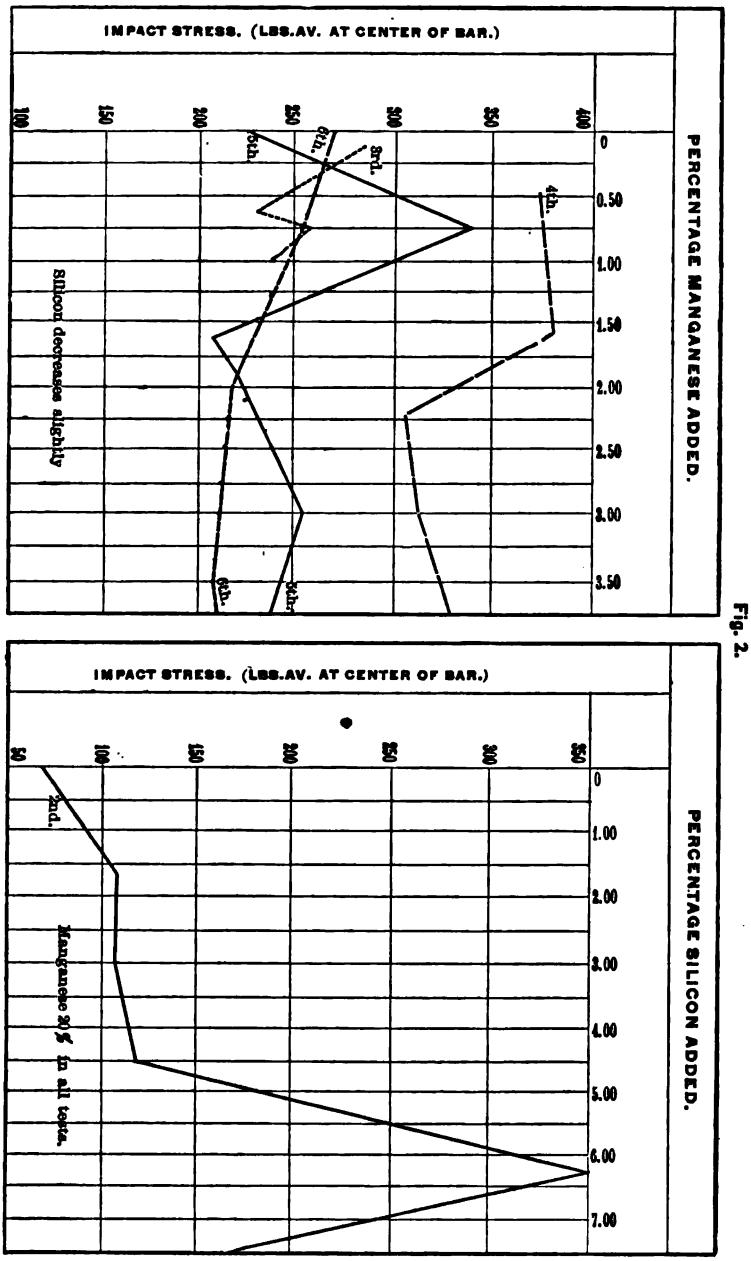


Fig. 2.

which we have the analysis of the pigs tested, and we find that the

shrinkage more nearly follows the variation of silicon than that of any other element. Manganese ranges from 0.01 in the lowest to 0.75 in the highest, and we cannot discover that between these limits in the pig, manganese exerts any influence, either for good or otherwise.

While we have already expressed the opinion that for a soft iron without shrinkage manganese should be absent, yet we do not think that where manganese in the pig is below 0.75, or even 1 per cent., its presence will ever be noticed. We think that the apparent increase or decrease in strength observable in some of our tests is due to other causes; for example, to some mechanical structure rather than to the presence or absence of manganese.

As will be seen from this paper it has seemed almost impossible to determine whether manganese was a benefit or an injury. It is only with the closest calculation and care that we have been able to determine the influence of manganese at all when present in cast-iron. Much of the manganese that is present in a pig-iron will escape during remelting, and it may aid in removing sulphur which has been brought in with the fuel.

In Figs. 1 and 2 the tabulated results of our tests are grouped and graphically presented, to show more clearly their general tendency.

Strength.—Fig. 1 shows the records of stress applied transversely and increased gradually, at the center of the test-bar of $\frac{1}{2} \times \frac{1}{2} \times 12$ inches; the record is in some cases the average of three, and in others represents one or two bars. A general glance at the figure conveys an impression that manganese does not influence the ability to resist this weight-stress. Adding manganese to molten iron generally reduces strength to resist such stress, as Series VII. and VIII. prove conclusively. These series were made at different parts of the year and each extended through at least seven days, during which time the cupola mixture varied more or less. In each of the fourteen casts the iron containing manganese was not as strong as that to which none had been added. We constructed Table VI. by beginning with the average strength of the cupola-iron on the first day of the test, which in Series VII. was 433, and the same day, with the addition of manganese, only 395, a difference of 38 pounds. The next day two casts were made, one with and one without Mn, and we recorded the difference. By subtracting each day this difference from the 433 of the first day we have a series of comparative records. The series taken as a whole shows, as did each day's tests, a decrease of strength as manganese increases. Series

VIII. was made in the same way and shows the same thing. In

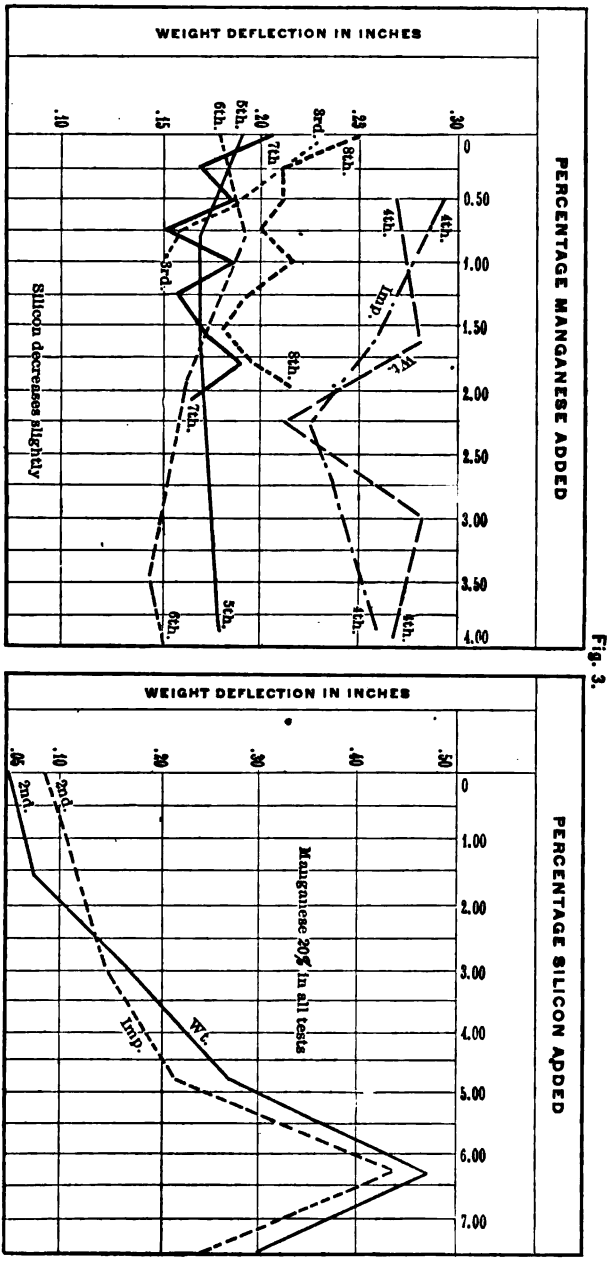


Fig. 3.

each of these the silicon was diluted with increase of manganese, and

ordinarily this would add to strength. No impact-tests were made of VII. and VIII. Series V. and VII. were each made so that the iron used was all taken from the cupola during one tapping, and we may, therefore, consider that the composition aside from the Mn additions is the same throughout. We had from each cast only two test-bars, one for testing by weight and one for impact. The records are not, therefore, as valuable as if we had been able to take the average of three bars. The record of impacts (Fig. 2) indicates a decrease of ability to resist blows or impact stress as Mn increases.

Series III. and IV. sustain the view already expressed. A review of Table VIII., in connection with the above, and with what we have said regarding our tests of various pig-irons, leads us to say that the small percentages of manganese found in commercial foundry irons will have little if any influence on strength. The records of the spiegeleisen series have no bearing on the case, except to show that, in the use of pig-iron, the increase of silicon in overcoming brittleness will entirely hide any influence manganese may exert. The additions of silicon along with Mn (Table VIII.) often cause an increase in strength.

Deflection (Fig. 3).—For Series II. and IV. we have plotted both weight and impact deflections. The other lines are all weight deflections. In each of these tests the decrease in silicon caused an increase of brittleness, and therefore lessened deflection; but the deflections of the spiegeleisen series and deflections in Table VIII. show that if silicon is increased along with manganese, the deflection may be greater with high manganese, or, as we may more safely say, deflection may be increased with a certain proportioning of silicon and manganese.

Set (Fig. 4).—The tendency to brittleness caused by the decrease of silicon in the tests plotted, and by the increase in manganese, has caused the iron to take less set, but the difference is not great. The uniformity of results causes us to believe this, and yet the remarkable increase in set shown by the spiegel series and the sets shown in Table VIII. lead us to state the fact as our results show it, and yet the effect is not marked enough to affect commercial irons with the ordinary percentages of manganese. Combinations of silicon and manganese may be made which will make cast-iron a very plastic metal. One of the bars in the spiegel series bent nearly one-half an inch in one foot and retained a set of one-quarter of an inch. Bars $\frac{1}{10}$ x 1 x 12 inches were bent cold in an ordinary tinner's rolls,

retaining a set of nearly two inches in a length of one foot; while

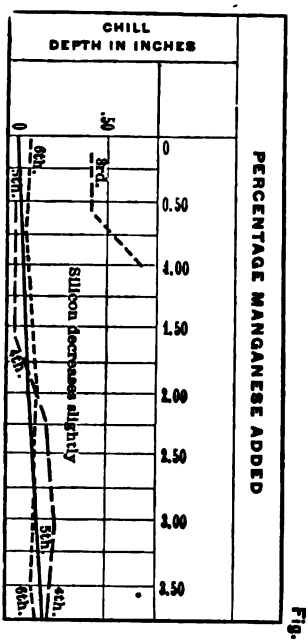


Fig. 6.

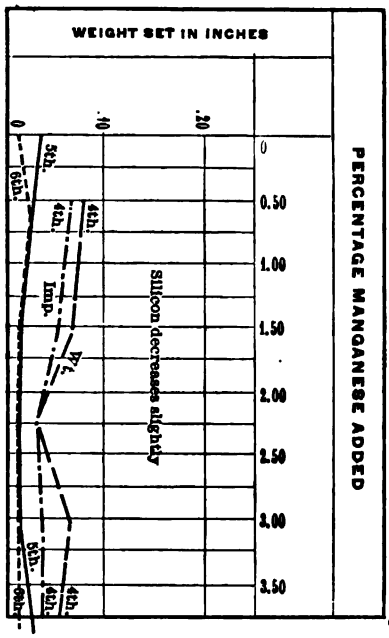
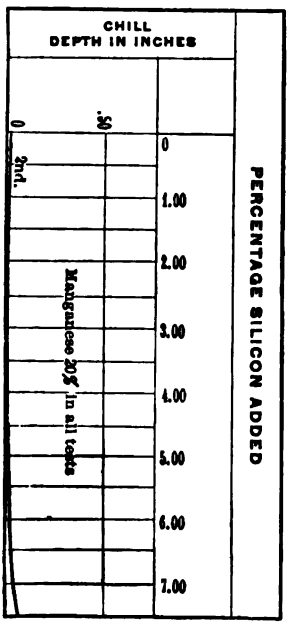
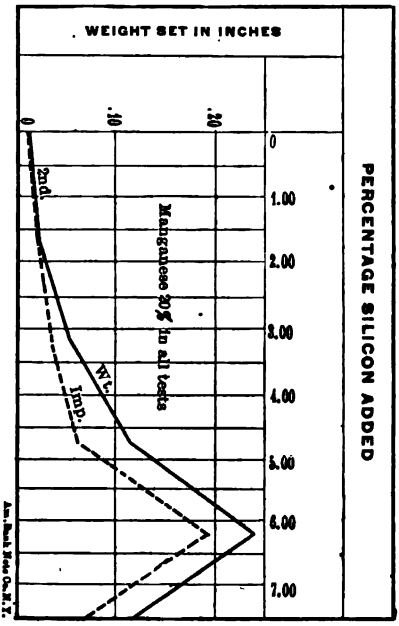


Fig. 4.





ordinary gray iron would take but little set before breaking.

Shrinkage (Fig. 5).—There is no room for doubt in this matter. Manganese increases shrinkage in every case (with the exception of IV.). Under the head of strength we have described the details of testing. Series VII. and VIII. are especially reliable. Table VIII. sustains the view that, with increase of manganese, shrinkage always increases. Much depends, however, upon how the manganese got into the iron. If present in small quantities in the pig it may not raise shrinkage as much as if introduced by a high-grade ferro-manganese. Moreover, in XIV., although 0.56 Mn is what would have been present if none escaped, yet in melting the pig-iron much of the Mn may have escaped. In XV. we took care that the 0.56 Mn should remain in the bars. We have found that the shrinkage which a pig-iron has when it leaves the blast-furnace will be difficult to take out. If an iron has a high shrinkage from excessive manganese this shrinkage cannot be removed unless the Mn is removed, and then it will not be as low as though the original pig had possessed a low shrinkage. Thus 20 per cent. spiegel shrinks 0.272 inch in one foot. Silicon reduces this to .213; and yet gray iron without Mn sometimes has shrinkage as low as .113. White iron shrinks about .260 to .240, and by silicon the shrinkage may be reduced to .140.

Melted wrought-iron has a shrinkage of about 0.220.

All of our tests and experience thus far show that shrinkage is increased by manganese.

Chill (Fig. 6).—Manganese is not hardened by sudden cooling and we should, therefore, expect that it would not add to the chilling quality of cast-iron. A decrease of silicon often increases chill, and this will account for the small increase in Series III., IV., V. and VI. If manganese has any chilling tendency the 20 per cent. spiegel series would show it, and yet each cast of that series shows absolutely no chill, except the one with 7.30 Si. It even seems to act in a contrary manner, as in irons without manganese we find it difficult to remove all the chill by additions of silicon; so the presence of manganese may possibly aid in removing chill.

In previous papers we have shown that an addition of silicon to white-iron reduces chill. Thus 3 per cent. of silicon reduced chill to 0.20, but further additions increased chill. With this white-iron we could not remove all the chill with silicon, but in the presence of 20 per cent. of manganese, even 3 per cent. of silicon removed all chill. The character of the white iron that held the 20 per cent. of Mn may have had something to do with this result, but we have

observed the same thing in various tests, and have also noticed that

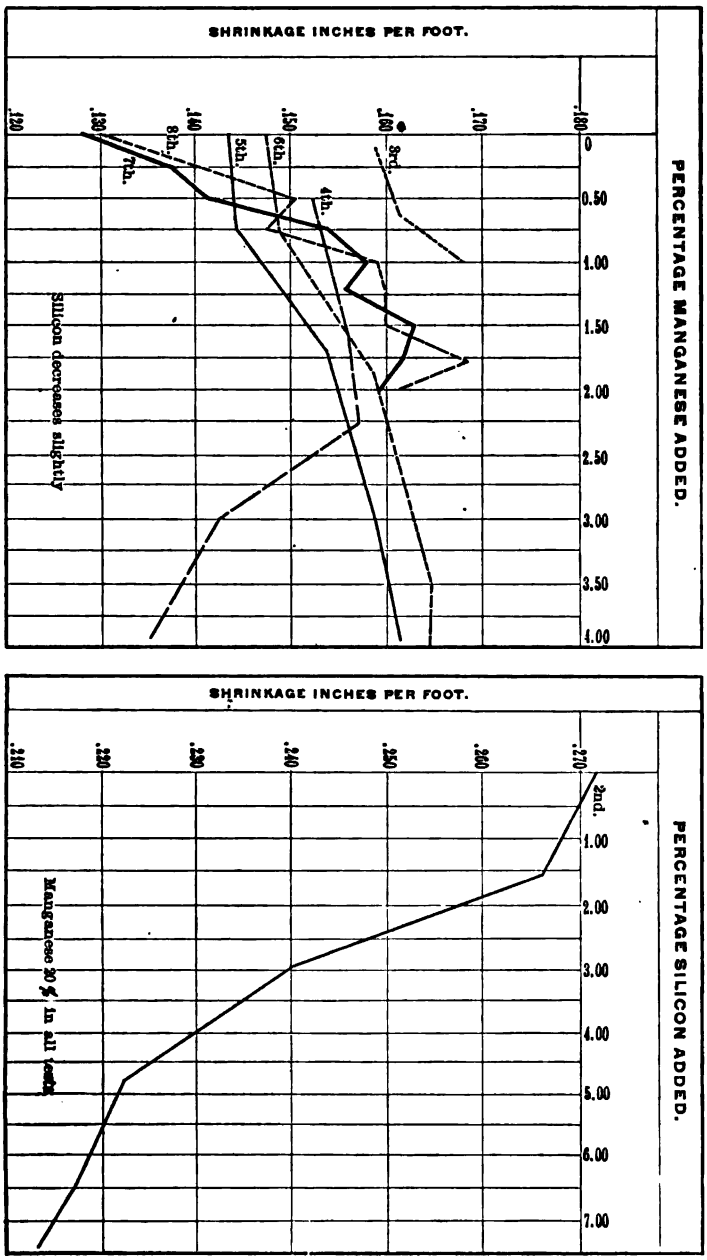


Fig. 5.

when manganese was present the chill did not stop off as suddenly

as when there was none, but the white and gray portions knit more into each other. If chill is due to a peculiar union of silicon and carbon, manganese may retard such union. But this is hardly proved by our tests.

Hardness (Fig. 7).—By an examination of hardness in each of the tables, it will be seen that manganese, almost without exception, hardens cast-iron. While we have shown that manganese does not increase the depth or whiteness of the chill, yet by increasing the hardness of a casting it also increases the hardness of the chill.

Grain.—Manganese being a white metal, is likely to lighten the color of a casting if introduced into the molten metal. When manganese is introduced into the pig-iron during its reduction in the blast-furnace, a larger percentage of carbon is taken up than if manganese were not present. The color then depends entirely upon the percentage of silicon in the iron. As silicon, by causing carbon to take the graphitic form, turns the iron to a gray color, the more carbon the iron contains the darker may it be if the right amount of silicon is present.

The more graphite and the slower the casting cools, the coarser is the grain. Manganese causes molten iron to cool more slowly, and to cool at the outside of a casting while the central portion is perfectly fluid. This slow cooling allows more graphite to form. A rigid shell having been formed, as the inner portion cools, it shrinks, and is not able to fill the interior, but leaves cavities. As the metal crystallizes, the graphite separates and lines these loose places. A casting or pig containing manganese is, therefore, likely to be coarse-grained and to lack solidity.

This is the appearance of the cast of 7.30 per cent. spiegel in Series II.; but while this metal seems soft, a file will hardly make an impression on it, and we could not cut with a cold chisel the piece that was left in the crucible. We were obliged to bring it to a white heat before the chisel could be driven into it with a sledge. Yet the grain was the same as that of any soft open gray iron.

The effect of silicon upon spiegeleisen has been noted before. Mr. R. A. Hadfield, in his paper on "Aluminum-Steel," read at the New York meeting of the Iron and Steel Institute (*Trans.*, xix., 1041), says that he added to 12 per cent. spiegel 3.73 per cent. of silicon, and turned the original specular white to a No. 3 mill iron. By adding 1.30 per cent. of aluminum to the same spiegel, it was turned to a No. 3 mill. (A specimen sent the writer by Mr. Hadfield is absolutely without chill.) He also added to a 25 per cent. spiegel

2.16 per cent. of aluminum with like results, only not quite so marked.

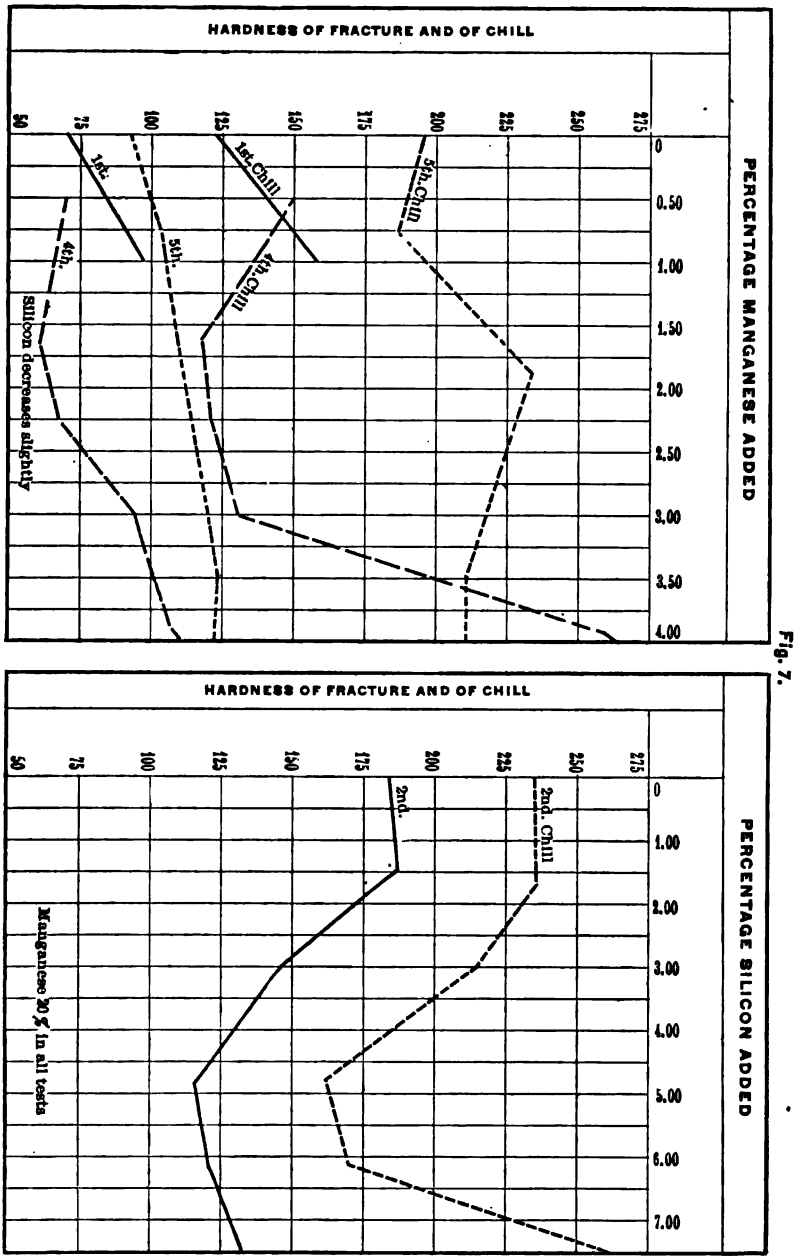

HARDNESS OF FRACTURE AND OF CHILL
50 75 100 125 150 175 200 225 250 275
PERCENTAGE MANGANESE ADDED
0 0.50 1.00 1.50 2.00 2.50 3.00 3.50 4.00
1st. Chill
5th. Chill
3rd. Chill
1st.
5th.
4th.
Silicon decreases slightly
HARDNESS OF FRACTURE AND OF CHILL
50 75 100 125 150 175 200 225 250 275
PERCENTAGE SILICON ADDED
0 1.00 2.00 3.00 4.00 5.00 6.00 7.00
2nd.
2nd. Chill
Manganese 20 % in all tests

Fig. 7.

This last experiment sustains our view that as Mn increases more silicon is needed to produce a given result.

Mr. Willard P. Ward, at a meeting of the American Institute of Mining Engineers, reported (*Trans.*, x., 268) that he made in 1875 a 16 per cent. spiegel perfectly gray and very tough and strong, and yet which could not be cut. Mr. Pourcel, at the next meeting of the Institute, reported (*Trans.*, xi., 197) a similar case in his experience of a gray 15 per cent. spiegel. His furnace, like Mr. Ward's, was very hot, and enough silicon was taken up to make the iron gray.

Discussion.

J. W. Langley, Pittsburgh, Pa.: I would like to ask Mr. Keep one question of a chemical nature, and that is whether in this line of work he has followed up the sulphur that is present. It is a very general impression that there is an antagonism between sulphur and manganese. I would, therefore, like to know whether he has found any relations between the increments of manganese and the decrement or otherwise of the sulphur in the metal?

Mr. Keep: It is, perhaps, to be regretted that all of the test-bars spoken of in my recent papers have not been subjected to chemical analysis. One reason is that I have no chemist at my disposal. The method I have pursued is explained fully in my paper. I think that my estimated composition of a given mixture of known materials is sufficiently accurate to warrant my general deductions. Of course, it cannot be as accurate as a complete analysis of the mixture itself.

Regarding sulphur; during my experiments, whenever I have put ferro-manganese into the ladle from our cupola, sulphur became apparent to the smell, which I presume indicates that the manganese eliminates the sulphur. It seems to me, moreover, that in Mr. Howe's work on steel, this is so conclusively proved as to make further experiment unnecessary. A curious and somewhat surprising experience is recalled by Prof. Langley's questions, namely, that we had at one time in our stock-yard a lot of pig-iron, made from cinder, which carried manganese and sulphur in almost the exact proportion of their chemical equivalents, *i.e.*, 27.5 to 16. The analysis was (Fleming) Si, 3.738; P, 2.329; Mn, 0.4395; S. 0.205.

With regard to manganese, my experience is that it is very difficult, by introducing manganese, as I have done, to get a series of bars containing just what is expected. In series IX. manganese was introduced with great care, and when I expected to find 2 per cent. I had only 0.583. That iron had been hot for several hours before

being cast. In other cases, I have reason to believe, there was no such large proportionate loss of manganese.

JULIAN KENNEDY, Pittsburgh, Pa.: Mr. Keep's remarks confirm the impression strongly made upon me by his paper, that, while the calculation of the contents of his mixtures might be in some cases approximately trustworthy, yet in the case of the high percentages of manganese it would be very misleading. We know that in running from the furnace to the pig-bed the composition of such metal is changed enormously; that 80 per cent. ferro-manganese will oxidize to powder unless completely protected from the weather. It is so sensitive that I should expect, in many cases, as much proportionate error as Mr. Keep has observed. I should not be surprised to find, as he did, that there was only one-half or three-quarters of one per cent. where two per cent. was expected; because the manganese is so extremely quick to oxidize. Another point that strikes me is, that it would be very interesting, in the case of those bars that were bent, to know the carbon contents. May it not be possible that the carbon had been pretty well driven out of these bars, and that there was in them what is known as Hadfield steel?

MR. KEEP: In the case of the bars of Series I., the manganese was not in the metal more than five minutes from the time that the melted iron struck the ferro. So that in that case we are pretty sure that all the manganese was still present. So far as the carbon is concerned, there was no indication on the surface of the casting but that carbon was still present. In the case of the specular bars of Series II., I put in 10 pounds of spiegel and melted it with ferro-silicon in the crucible. In that case there may be less manganese in the bar than was in the pig, but Mr. Hadfield's analyses of similar tests (*Trans.* xix., 1041) show that enough Mn remains to justify our conclusions.

PRESIDENT BIRKINBINE: I think we can congratulate ourselves upon such papers as Mr. Keep's, not only because of their value technically, but also as showing that the men who use pig-iron are beginning to investigate what it is. We all know that foundry pig-iron is graded differently in different parts of the country, so that the names given to the various grades have nothing more than local significance. The Bessemer steel-makers have done a great service to our blast-furnaces by practically ignoring mere names, and classifying irons according to their exact composition. Now the foundrymen, by beginning to investigate thoroughly the material which comes into their yards, and not accepting or condemning it on its

name alone, are taking a great step in advance. A pig-iron made from a furnace-charge containing a little mill-cinder may be better than if an ore worse than mill-cinder were substituted for that material. The true test is the analysis and the experimental use of the pig for the special purpose in view.

E. C. PECHIN, Roanoke, Va.: A number of years ago, first at Pittsburgh, and afterwards at the New York meeting, Mr. John L. Gill, Jr., of Pittsburgh, claimed that the chilling property of certain pig-irons depended, not absolutely upon the proportion of silicon, but upon the presence of a form of carbon intermediate between the combined and the graphitic iron. I have never seen anything more about it, and I simply ask, for information, whether anybody else has discovered this intermediate form of carbon.

ELECTRICITY IN MINING, AS APPLIED BY THE ASPEN MINING AND SMELTING COMPANY, ASPEN, COLO.

BY M. B. HOLT, ASPEN, COLO.

(Glen Summit Meeting, October, 1891.)

AT this time, when electricity in its various applications is attracting so much interest, and with such good reason, it has been suggested to me that, as the Aspen Mining and Smelting Company, of Aspen, Pitkin county, Colorado, was the first mining company in America to employ electrical power for hoisting, a record of what has been accomplished by this company would be of interest.

Early in the year 1888 the question of power for the development of the ore-horizon lying below the level of the tunnel through which the property is worked, became one of importance. The conditions under which it was proposed to use power were as follows (See Fig. 1): The company's mines lie upon the northwestern slope of a mountain, the ascent of which is at an angle of 28° from the horizon. The ore, consisting of argentiferous galena and other silver-bearing mineral, is found at or near the plane of contact of a superincumbent mass of blue limestone, with an underlying bed of dolomite. This contact plane dips north 30° west, at an angle of 60° with the horizon.

A tunnel 1000 feet in length, driven south into the mountain near

its base, penetrates this ore-mass at a vertical depth of nearly 500 feet, and through this the ore and the waste material are brought to the surface. For the purposes of drainage and transportation, the tunnel has received a grade of 3 per cent. towards its mouth, while

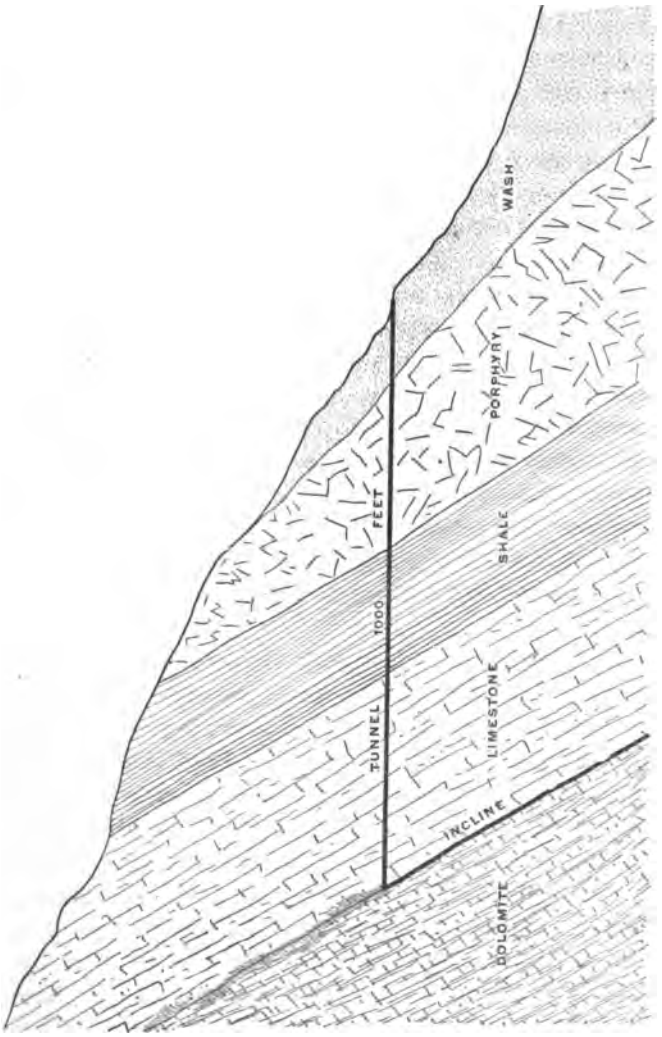


FIG. 1 Cross-section, showing Situation of the Aspen Company's Mines.

from its breast, or inner extremity, level branches or drifts are run both to the right and left, following the strike of the ore-bearing plane, and developing it in its lateral extent. The development below the tunnel-level was begun by sinking two inclines, which were

intended to follow the ore-plane in its descent, and which, it was contemplated, might acquire considerable depth. One of these inclines began near the inner extremity of the tunnel, and the other was started 300 feet distant in the drift toward the north. Both descended below the tunnel-level at an angle of 60°. Power, applied to hoisting machinery of some kind, was necessary to bring up the ore and waste material from these inclines, and from the levels that would be driven in future.

The first plan considered was the use of steam-power; that of compressed air was then discussed. Both of these plans were finally laid aside for weighty reasons, although the latter offered several advantages over the use of steam in the inside workings. It then occurred to Mr. Fred. G. Bulkley, the manager of the company, to duplicate the operations of the Compagnie de la Ferronière, on a larger scale; to use electrical power, utilizing, for the generation of the current, the power daily going to waste through the channels of mountain streams in the vicinity. Accordingly a flume was constructed, with ample capacity, giving an effective head of 63 feet, and a Pelton water-wheel, having a diameter of 5 feet, was selected for the motor. A 50-horse-power 500-volt constant-potential Edison dynamo was then set up and run under the direction of the Roaring Fork Electric Light and Power Company. This machine generated the current at a distance of 6000 feet from the mouth of the tunnel.

After long delays and by constant importunity, attended with more than one failure, Mr. Bulkley finally secured the attention and aid of a manufacturer of electrical machinery, to provide the hoisting-plant for the mine, and finally, in July, 1888, the first electric-hoister used in mining work in America was placed at the head of the Veteran Tunnel, and successfully used for hoisting from the incline near this point. The same motor, by engaging with a second drum, draws the empty cars in the tunnel by a tail-rope system, the loaded train running out by gravity. This hoist consists of a 7½-horse-power street-car motor (See Fig. 2), manufactured by the Sprague Electric Railway and Motor Company, geared as shown in Fig. 3. This motor proved the success of the plant from the start, notwithstanding the trying conditions under which it was at first used—one of which was the great amount of moisture in the workings, which condensed upon the machine. Later, the iron pinions on the armature-shaft were exchanged for raw-hide pinions, diminishing the noise when running, and also the liability of any leakage of

current to the levers and other parts of the machinery. The working of this motor was so successful that during the same year a similar hoist was placed at the head of the incline 300 feet to the northeast. Each of these hoisters was designed to raise 250 tons

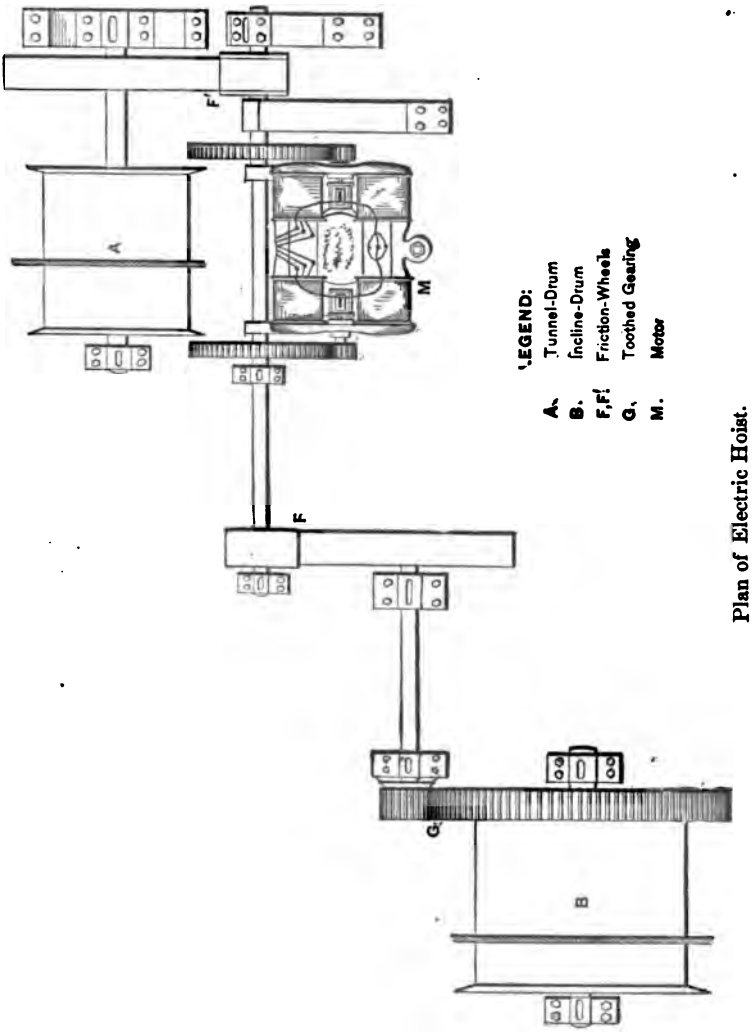


FIG. 3. Plan of Electric Hoist.

250 feet up a 60° incline every 24 hours, estimating 16 hours of actual working time. Fig. 3 shows, in plan, the arrangement of the electric hoist.

The first motor has been in constant use for hoisting for 3 years, during which time the cost of repairs has been as follows:

Replacing field-magnet coils,	$43.00
Re-winding one armature,	80.00
Replacing commutators,	75.00
Total repairs,	$198.00

This does not include the cost of commutator-brushes, which have to be replaced from time to time—which is, however, a small item. The first item of expense above given was necessitated by the burning out of the insulation on the wires of the magnet-coils, which might have resulted from one or more of several causes. The second item was due to the bursting of the binding-wires on the armature. This, had it been anticipated, might have been guarded against. The last item, caused by the wearing out of the commutators, was due to the natural wear between the commutator and the copper brushes which were originally employed.

Recently the company has put carbon brushes upon all of the motors, and also uses them upon the dynamos. Since this change has been made, the motors have run without sparking, and the wear upon the commutators is inappreciable.

The Company has an extra armature that can be used in either motor-hoist, so that, in case of accident, very little time need be lost in repairing the damage. In the case of the accident to the field-magnet coils, mentioned above, the burnt coils were replaced without much delay by others bought in Aspen. The total time lost, due to accidents or repairs during the time the hoist has been in use, amounts to 12 hours.

In the latter part of the year 1889 the Company, having available water-power, not in use, decided to have its own generating-plant. Previous to this it had been supplied with power from the Roaring Fork Electric Light and Power Company. Work was soon begun, and by July, 1890, the Company was generating its own power. This plant has a capacity of 100 electrical horse-power, which is generated by two 50-horse-power Thomson-Houston motor-type dynamos, wound for a constant potential of 500 volts. These are run from two double Pelton water-wheels, 42 inches in diameter, *i.e.*, four 42-inch wheels, two being paired together on each shaft. Each of these four wheels is impelled by the force of the water issuing from two nozzles and applied to the wheels, as shown in Fig. 4.

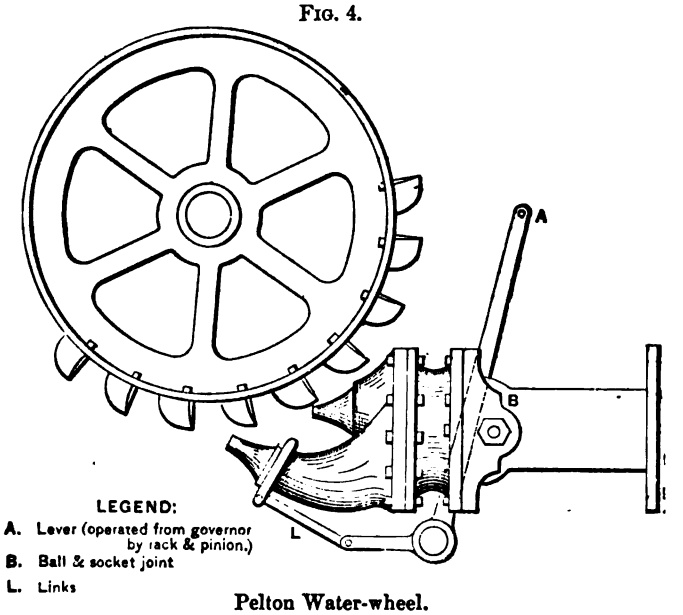
A pressure of about 35 pounds per square inch, as indicated by a gauge at the foot of the main column-pipe, is obtained from an effective fall of a little over 80 feet. This fall is obtained by fluming

for a distance of 1300 feet. The flume has a capacity of 1000 cubic feet of water per minute.

The requirements calling for greater or less power are met by deflecting the nozzles, which work on a ball-and-socket joint, by means of a Woodward governor, set to maintain a constant speed of the water-wheel shaft from which it is actuated. The wheels are enclosed in an iron hood, 4 feet by 4 feet, which can be taken off with little trouble, the joints being kept tight with rubber gaskets. (See Fig. 5.)

The adjoining ends of the water-wheel shafts carry a friction-

FIG. 4.



Pelton Water-wheel.

clutch, by means of which the shafts can be run as one shaft when it is desired to throw the output of both dynamos on one common line.

Fig. 6 shows one set of water-wheels and one dynamo, with the Woodward governor, the other set of wheels and dynamo being entirely similar.

The electrical connections in the generating station are as simple as possible, the dynamos being connected with the switch-board by overhead wiring hung from the trusses.

Wiring.—The generating station is situated 6000 feet from the entrance to the tunnel. The stationary motors now in use are situ-

ated at distances of 1800 feet, 1200 feet, and 1000 feet from the entrance underground. From the power-station to the tunnel the current is carried by bare 00 copper wire, except for a distance of about 300 feet at each end of the line, where an underwriters' insulated wire of the same size is used. Inside the mine the current is carried to the two main hoisting-stations at distances of 1000 and 1200 feet by kerite seven-strand conductors, having a heavy insulation $\frac{14}{32}$ inch in diameter. Okonite or Grimshaw insulated wires are used on all the other circuits inside the mine, where the insulation must necessarily be of the best. On none of the circuits inside the mine is there a loss exceeding 5 per cent. Along the outside circuit there is at present, when the wires are carrying the maximum load required by present operations, a loss of from 5 to 6 per cent.

Cost of Generating.—The cost of generating, as obtained by dividing the cost of labor and lubricants (interest and depreciation are not included) by the horse-power demanded, amounts at present to two-thirds of one cent per horse-power per hour; and greater amounts of power could be furnished at a lower rate per horse-power, as up to 100-horse-power no increase of plant would be necessary, the cost for labor would remain the same, and the cost of lubricants would advance but little.

In July, 1890, the Company received from the Sullivan Diamond Prospecting Company, of Chicago, an electric diamond prospecting-drill. (See Fig. 7.) This was the first drill of its kind, and was built for the Company at the request of Mr. Fred. G. Bulkley, with the understanding that it should have a thorough trial, and be retained if satisfactory. It is operated by a 3-horse-power Thomson-Houston motor, and has a capacity of 400 feet, taking out a core $\frac{15}{16}$ of an inch in diameter. The drill was first set up on the surface, and experiments were made with it by drilling into an immense granite boulder. It was found that, in this kind of rock, it would drill, taking out a core of the above diameter, 2 inches per minute. It was then put in the mine, where it has been used steadily for the last 12 months for exploring the hanging- and foot-walls of the mineral deposit, which are respectively a hard blue and often silicified limestone, and a brown dolomitic limestone. In these rocks the machine drills at the rate of $1\frac{1}{2}$ inches to 2 inches per minute; or, in a shift of 8 hours, allowing for all delays essential to diamond-drill work, the progress made varies from 6 to 32 feet, depending upon the character of the rock. The average per shift, allowing for time lost in moving, setting up and drawing cores, is 15 feet, one

drill-man and one helper being employed. About 4400 feet have been drilled to date of writing, July 6, 1891. The exact cost, including all expenses, has been 68 cents per foot. This is rather higher than was expected, and it is thought that it can be reduced.

Recently finding that the 7½-horse-power hoist at the head of one of the inclines was getting too small for the work required, it was removed to another part of the mine, where it could be advantageously employed, and a 25-horse-power C. & C. electric motor put in its place. This larger motor is geared so as to raise a load of 3000 pounds up a 60° incline 275 feet per minute. It is capable, by actual timing, of making the round-trip from a depth of 550 feet, with a load of 3000 pounds, in 3 minutes.

At the time of writing, the Company is about to apply electrical power for running a small machine-shop and its timber-room and carpenter-shop. It is also proposed, in the near future, to employ an electrical locomotive for surface-tramming.

A contract has been made with the Thomson-Van-Depoele Electric Mining Company by which this Company is soon to receive, upon trial, a Thomson-Van-Depoele electric percussion-drill, and a motor-transformer for supplying the drill with a current of 220 volts from our 500-volt mains.

The main working-galleries of the mine are lighted by electricity, the current for which is taken from the power-mains, and five 100-volt incandescent lamps are connected up in series. As the light produced is found to be sufficient, and the life of the lamp is thereby prolonged, six of these lamps are, in most cases, placed in series with each other. Eight horse-power of electrical energy is now employed for lighting purposes.

The recognized advantages of electrical power for mining operations may be briefly summarized as follows:

1. It can be transmitted long distances with small loss, thus making it possible to use power at such a distance from its source as would render it otherwise unavailable, as in the case before us.

2. The conductors for conveying electrical power from one point to another require less space, are more easily put in place and repaired, are easily tapped for branch circuits, and form a more flexible system throughout than any other mode of transmission permits.

3. The electrical system is ideal, viewed from the standpoint of cleanliness.

4. The stations for utilizing electrical power can be made to occupy a minimum space.

5. If this system does not assist ventilation, it does not, on the other hand, vitiate the air in the mine-workings.

After three years' use under the varying conditions of mining work, the electric current of 500 volts has proved itself free from danger to life, and has caused no inconvenience further than one or two severe shocks. Even the risk of receiving a shock has proved insufficient to suppress the spirit of investigation with which some men are endowed. The best illustration of the convenience and flexibility of the system is the diamond-drill, where the conductors are unwound and strung up as the drill moves along, or taken down and coiled up, as may be desired.

A CHINESE SYSTEM OF GOLD-MILLING.

BY HENRY LOUIS, ASSOC. R.S.M., F.I.C., F.G.S., ETC., SINGAPORE, STRAITS SETTLEMENTS.

(Glen Summit Meeting, October, 1891.)

THE object of the present paper is to describe a primitive method of gold-extraction, practiced by a small colony of Chinese in the district of Tomoh, one of the Siamese-Malayan States. This district has been worked for gold about 150 years, but has been only twice or thrice visited by Europeans, who are looked upon with a good deal of jealousy by the miners.

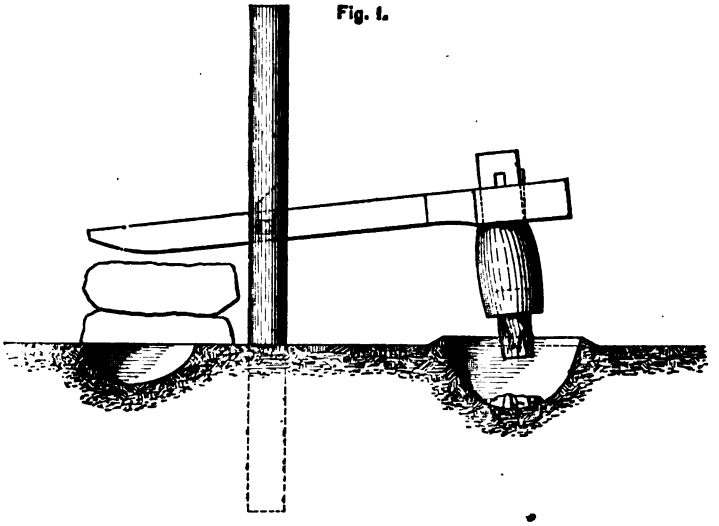
The gold-district, covering not more than 10 square miles, is situated about Lat. 5° 47′ N. and Long. 101° 39′ E., on the headwaters of the river Teluban. It is a very mountainous region, lying on the northern flanks of the great range that forms the watershed between the rivers Patani, Kelantan, Teluban, etc., on the north and Perak on the south.

The gold occurs in narrow veins and leaders of quartz, intersecting and intercalated among irregularly upheaved and contorted, highly metamorphosed, micaceous and chloritic schists. These veins vary between 3 inches and 3 feet in thickness, and are sometimes very rich. The geology of this region is thus very similar to that characterizing gold-regions all over the world; it must, however, be

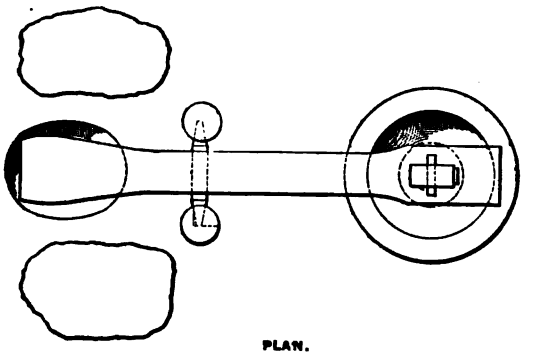
noted that, owing to the extreme heat and moisture of the climate, the rock has in many places weathered very completely into soft red, purple and yellow clays, such decomposition extending often to great depths.

The history, also, of this remote gold-region is curiously like that

Fig. 1.



ELEVATION PARTLY IN SECTION



PLAN.

FOOT-POWER QUARTZ-CRUSHING MILL, TOMOH.

Scale 1 inch = 2 feet.

Am. Bk. Note Co. N.Y.

of other similar ones. Gold was first discovered by Malays in the sands or banks of the river Teluban; and they perhaps worked it to some small extent. The Malays are, however, constitutionally averse to continuous work in any shape; and before long Chinese immigrants, steady and inured to labor, came into the country and

"rushed" these alluvial gold-fields. The Malays every now and then killed off these immigrants, partly for the pleasure of the deed and partly to gain possession of the gold they had washed out; but with characteristic perseverance the Chinese returned again and again in ever increasing numbers, and have now a firm footing in the country, the Chinese population of the Tomoh district being at at present over 1000.

They seem to have worked their way up the river from one alluvial field to another—each being limited in area, though some are very rich—until, the alluvials being exhausted, they had to turn to the reef-gold of the above-mentioned Tomoh district, which they began to attack some 70 years ago.

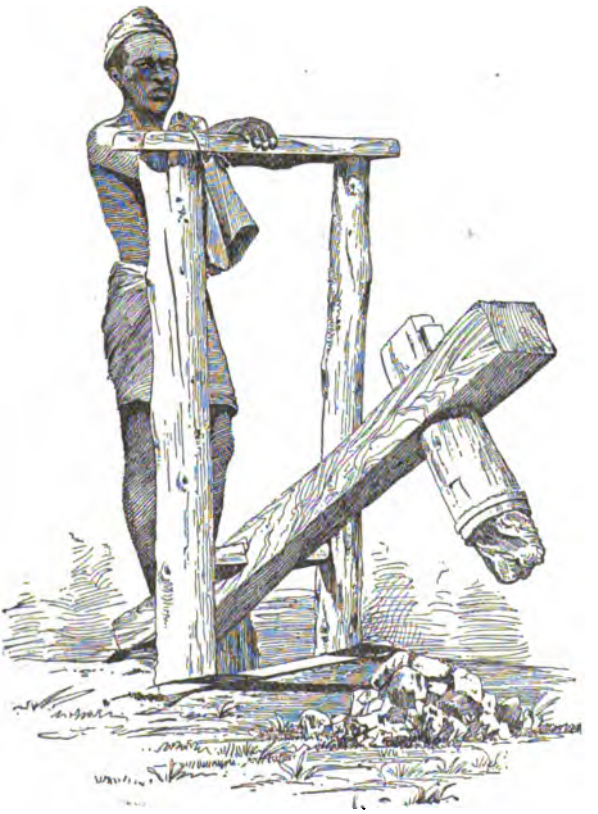
Their method of mining was then, and is now, the following: A small water-furrow is first brought in at the highest possible level on a suitable hill-side, and the stream is turned down the hill. By means of a heavy long wooden crowbar, shod with a long strongly-made chisel-pointed iron socket, and with the help of the stream of water, which rarely exceeds 50 cubic feet per minute, the surface-soil and weathered country-rock are loosened and sluiced away. No trouble is taken to save any of the gold washed down, except in one or two instances where rude riffles have been inserted in the tail-race; the race is, however, carefully searched for bits of quartz showing visible gold, which are picked out and put on one side. The surface of the shales is thus stripped, and any veins of gold that may be laid bare are then worked. The principal mining-tool is a rough kind of pick, and the use of explosives, or even of wedges, is quite unknown. Neither shovels nor barrows are used; their places are taken by broad hoes and baskets, a pair of the latter, swung at each end of a stick and holding at least 70 pounds, being easily carried up steep grades by a Chinese miner. The tunnels, small and irregular, usually incline steeply upward; they are rudely timbered, and as timber decays rapidly in this climate, these workings cannot penetrate far into the hills, but soon have to be abandoned, and the whole series of operations has to be recommenced.

A party of 27 miners, who owned and worked a rich hill-side, considered themselves to be doing well when their entire day's output (they do not work night-shifts as a rule) was a little over half a ton of quartz. The quartz, as extracted from the reef, is cobbed down with hammers to about pass a 1½-inch ring, and is then carefully hand-picked, all stone showing visible gold, sulphurets or any other favorable indications being sent to the mill and the rest

being thrown away. From one-eighth to one-half is thus rejected. I have assayed many samples of this refuse rock, which carries from 3 to 10 pennyweights of free milling gold to the ton, so that it is quite worth milling according to our modern ideas.

At first the mode of crushing adopted by the Chinese consisted in heating the rock red-hot, quenching it in water and then pounding it down and rubbing it between two stones. About 35 years ago a

FIG. 2.



Operation of Foot-power Mill.

tilt-hammer, made entirely without iron and having a stone head, was introduced, and is still much used by individual miners. About twelve years ago the battery of three to six hammers, worked by a water-wheel, was first employed. It is said to have been copied from mills for crushing the materials of "joss-sticks." Tilt-hammer rice-mills are also built. Such water-mills are usually the property of a party of miners working together.

I propose to describe the method of milling with both the foot- and the water-mill, a typical pattern of the former being shown in Figs. 1 and 2 and of the latter in Figs. 3 and 4.

The foot-mill shown in Figs. 1 and 2 is of the usual type, from which there are but few unimportant departures. The entire falling weight is about 45 pounds, and the length of drop about 20 inches; as a rule, these mills are worked at 15 to 20 blows per minute.

The mill shown is built entirely without iron; the stone that forms the base of the mortar is a piece of hard quartzite or of barren reef-quartz, the same material being used for the hammer-head, which is firmly held in its socket by wooden wedges, the socket being kept from splitting by a stout hoop of rattan twisted round it. Some of the mills use iron hoops, and some have iron spindles for the hammer to work on; with these exceptions and one or two other very unimportant details, the construction is always the same, though the dimensions may vary a little. There is scarcely a house in the whole district that has not one of these mills.

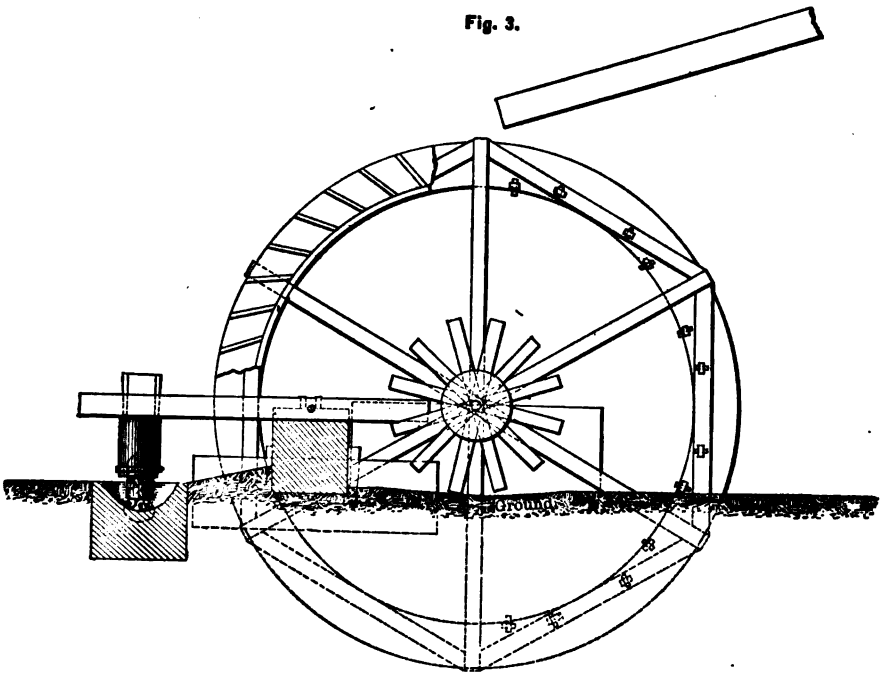
The Chinese usually work these mills for about eight hours per day. A shovelful of quartz is first thrown into the mortar and the mill is then worked by the foot of the miner, who stands on one or other of the stones shown in the drawings, grasping the uprights or else a cross-bar that is sometimes fastened across them.

When the quartz is supposed to be crushed sufficiently fine, the hammer-head is propped up, and the crushed stone is scraped out and sifted through a circular sieve 15 inches to 20 inches in diameter, and about 1½ inches deep. The sieve itself is made of thin strips of rattan about 0.1 inch in width. There are from 36 to 40 holes per square inch, so that the width of mesh varies between 0.04 and 0.06 inch. A man can crush in a working day, with one of these mills, from 70 lbs. to 140 lbs. of stone, according to its hardness.

The number of heads in a power-mill varies between 3 and 6, depending principally on the quantity of water available. As the district is well watered, the large majority are 6-stamp mills; out of 11 power-mills which it contains, 8 are 6-stamp mills. Figs. 3 and 4 show the usual type of the latter mills, from which pattern there is practically no departure. I could not even induce the Chinese to try a curved cam instead of a straight one, as they seemed to consider such innovations dangerous; and they added that "wood and water were both cheap enough." As will be noticed, the construction of the water-wheel is extremely crude—the water, which is

sometimes brought down very steep hills from considerable heights in small, highly-inclined ditches, strikes the flat buckets with considerable velocity, so that the wheel is partly an impact and partly a pressure wheel; the buckets are never more than half-filled at the best, and the wheel is sometimes allowed to wade in tail-water to the full depth of the shrouding. Much power is accordingly wasted, the amount of water consumed in driving one of these mills being

Fig. 3.



ELEVATION PARTLY IN SECTION.

WATER-POWER QUARTZ-CRUSHING MILL, PACHO NEAR TOMOH.

Scale ½ inch = 2 feet.

Am. Bk. Note Co. N.Y.

from 80 to 100 cubic feet per minute. The average number of drops of each head varies between 27 and 32 per minute; the length of the drop is about 2 feet, and the effective falling weight of the head is about 70 lbs. Thus only about one-third of the theoretical power of the water is utilized, but of course much of this loss of energy is due to the friction of the whole machine, notably between the straight cam and the tail-piece of the hammer. There are usually 3 men per shift working one of these mills, 2 being engaged in looking after and feeding the machine, while the third sifts the

pounded stone as already described, throwing back under one of the hammer-heads whatever will not pass the sieve.

The cost of one of these mills complete, including a substantial shed over it thatched with palm leaves, but excluding the water-furrow, is said to be about $2000*, and they are supposed to last from 5 to 7 years—needing, however, constant repairs.

A stone hammer-head lasts from a week to a month, according to its quality. They are made, as in the foot-mills, from boulders of quartz rock, and it is mostly one man's business to search for these boulders in the bed of the stream, and, when found, to dress them into shape.

The crushing capacity of one of these mills varies from 850 lbs. to 1400 lbs. per 24 hours, according to the hardness of the rock. It is to be specially noted that the crushing is performed quite dry.

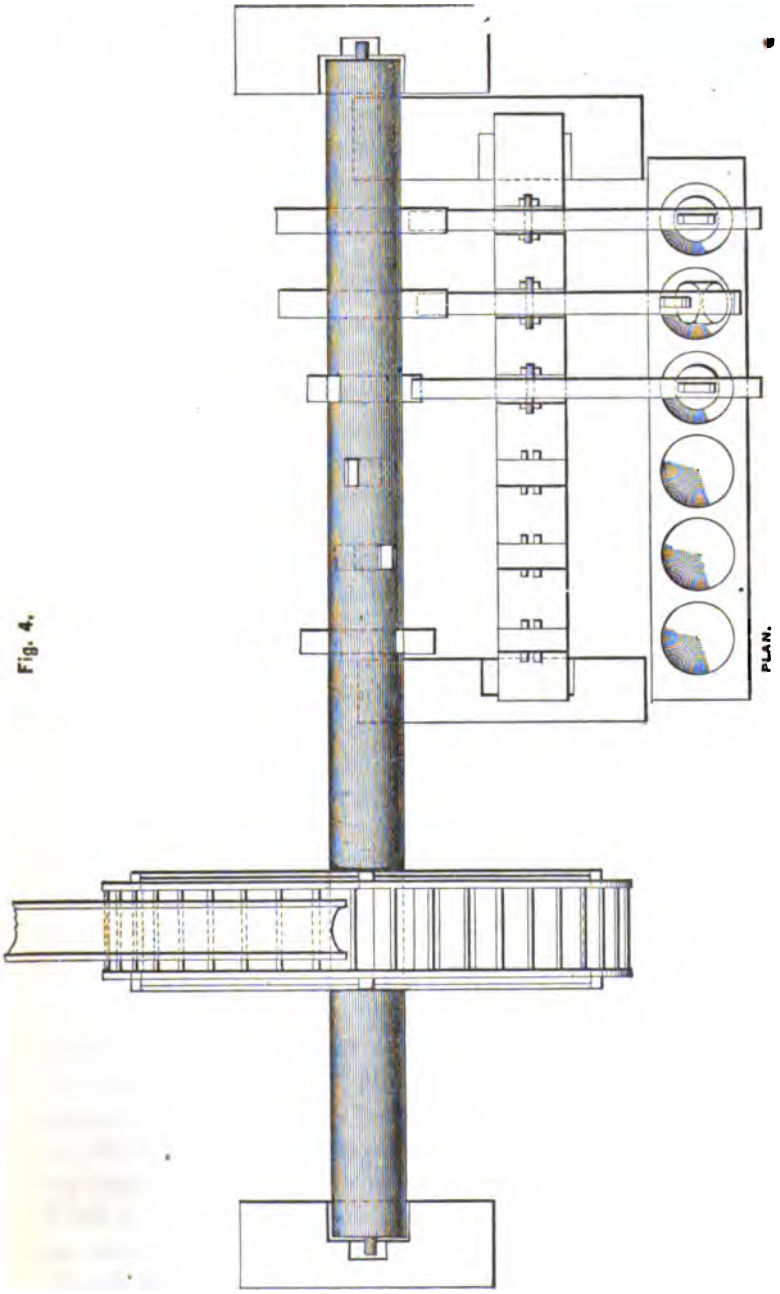
I tested the degree of fineness to which these mills reduce the quartz by differential siftings of a number of samples, taken by spoon-sampling the heaps of crushed ore lying at various mills. The results of some of my tests are given in the following table:

Size of holes in original Chinese sieve—inches square.	0.042.	0.055.	0.055.	0.062.	0.062.	0.052.	Average.
Left on sieve with holes 0.050 in. sq.	0.9	0.2	1.0	0.5	0.5	0.6	0.6
Passed through " " " " "	19.8	21.5	18.7	24.0	20.8	20.3	20.8
" " " " 0.024 " "	14.3	15.9	16.8	15.8	13.7	12.0	14.8
" " " " 0.018 " "	12.6	13.5	17.0	16.2	16.9	11.4	14.6
" " " " 0.011 " "	6.1	10.9	8.1	7.5	16.1	6.0	9.1
" " " " 0.008 " "	46.3	38.0	38.4	36.0	32.0	49.7	40.1
	100.0	100.0	100.0	100.0	100.0	100.0	100.0

It appears from the above table that a great deal of the ore is crushed very fine (too fine, indeed), while some is not fine enough. As about 40 per cent. of the ore will pass through a 6,400 sieve, there must be much over-stamping, resulting, no doubt, in the production of a great deal of float-gold and slimes.

After the mill has been running for a longer or shorter period, according to circumstances, a clean-up takes place. The crushed ore is carried out in large wooden pails to a Chinaman, who washes

* The dollar here referred to is the old Spanish dollar, worth about $0.87 in United States money.



PLAN.

Fig. 4.

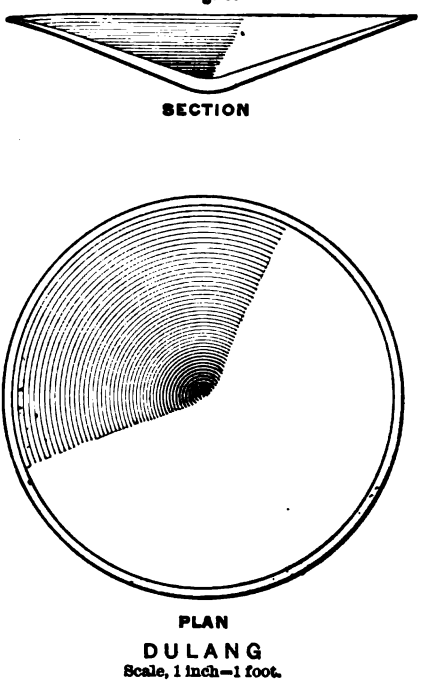
WATER-POWER QUARTZ-CRUSHING MILL, PACHO NEAR TOMOH.

Scale ½ inch = 2 feet.

it, squatting down by the side of a square pit, through which a small stream of clear water is kept running. The implement used for washing is a flat, somewhat conical wooden dish, cut from the spurs of certain hard-wood trees, and fashioned with much care. It is known as the *dulang*, and much resembles the Spanish-American *batea*, except that the section of the former is that of a very obtuse rounded cone, while the section of the latter is approximately that of a sphere.

A section of a typical *dulang* is shown in Fig. 5. Much impor-

Fig. 5.



DULANG
Scale, 1 inch=1 foot.

tance is attached to the correct shape of the conical point, as it is in this that the precious metal is gathered together. The *dulang* is filled with from 10 to 15 lbs. of crushed stone, according to its size, and this is washed by a curious circular, combined with a slight undulatory motion, by which the particles of light, barren quartz are swept over the edge of the *dulang*, which is held just dipping below the surface of the water in the pit, while the heavier particles are collected in the rounded apex of the cone. When nearly cleaned, the gold and concentrates are transferred to a smaller, very carefully made and polished *dulang*, about 1 foot in diameter, in which the

quartz is washed off as thoroughly as possible, and the gold, by a skillful jerk, is thrown clear from the sulphurets, and finally collected in a small brass dish. The sulphurets still retain much coarse gold, to which they cling obstinately. They are ground as fine as possible on a stone and re-washed several times, a good deal of the gold being thus separated and added to that previously obtained. Even then the sulphurets still carry much gold, the larger portion of which is free. They are stored away in jars while wet and allowed to rust, and after a time they are sometimes re-crushed and re-washed; very often, however, they are merely allowed to accumulate and are not treated further. The first tailings are re-washed, and then stacked.

The cleaned gold is dried and melted over a small forge provided with a box-shaped wooden blower of the usual Chinese type. The fuel is charcoal. Tiny, conical crucibles, capable of holding about a couple of ounces of gold are used; the gold-dust is melted in these with borax and niter as fluxes; the slag is lifted off the surface of the gold when the latter is supposed to be clean, by means of an iron rod, and the gold is then granulated by pouring into water. If it is not considered to be sufficiently soft and pure it is re-melted, and the process is repeated until the gold is quite soft. The principal impurities removed seem to be sulphur, arsenic, a little copper, and perhaps traces of lead. Both the granulated gold and the crude gold-dust, as also gold got from river-washing, are used as currency in this district, coined money being scarcely ever seen here, and then only in the form of the old Spanish dollar.

In a partial wash-up at one of these mills, during my stay in the district, the following results, considered to be exceptionally good, were obtained, the quantity washed being as nearly as possible 2000 pounds of crushed ore:

	oz.	dwts.	grs.	U. S. currency.
Rough gold, before melting, . . .	3	11	7	worth $52 60
5½ pounds of sulphurets for retreatment, .	0	16	0	" 14 75
28¾ pounds of sulphurets supposed to be cleaned, ...	0	6	0	" 5 60
Total gold, about, ...	4	13	7	" $72 95

The above two parcels of sulphurets gave by fire-assay respectively, per ton of 2240 pounds:

	I. Rich.			II. Poor.		
	oz.	dwts.	grs.	oz.	dwts.	grs.
Gold, ...	293	9	13	21	5	8
Silver, ...	30	7	3	2	1	16
Total, ...	323	16	16	23	7	0

The tailings from this (or a similar) washing gave by fire-assay, per ton of 2240 pounds:

	oz.	dwts.	grs.
Gold,	0	15	16
Silver,	0	7	12
Total,	1	3	4

As a general rule, there seems to be left in the tailings about one-third of the gold originally present in the ore, while there must be a considerable additional loss of float-gold carried away in the process of washing, due to the original fineness of some of the gold in the ore, and to the over-stamping already referred to.

The Chinese themselves seem to be of opinion that they get about one-half of the gold originally contained in the stone.

The following examples will show how much gold is retained in the tailings:

In working a very rich ore, which assayed, per ton of 2240 pounds,

	oz.	dwts.	grs.
Gold,	6	12	15
Silver,	1	1	9
Total,	7	14	0

the tailings, after three times pounding and washing, still assayed:

	oz.	dwts.	grs.
Gold,	2	16	10
Silver,	0	5	6
Total,	3	1	16

thus showing that nearly one-half of the proportion of gold originally present was still locked up in the tailings.

In another instance, where the quartz assayed originally:

	oz.	dwts.	grs.
Gold,	1	4	6
Silver,	0	3	1
Total,	1	7	7

two piles of tailings, both said to have been exhausted by five or six poundings and washings, assayed respectively:

	dwts.	grs.	dwts.	grs.
Gold,	4	15	11	1
Silver,	0	14	1	12
Total,	5	5	12	13

From the average of these two assays it would appear that nearly one-third of the original proportion of gold is still left in the tailings. I might quote numerous other assays, but the results in all cases were approximately the same; there were no really clean tailings at all, in spite of the fact that they were all the result of handling surface-ores, where practically the whole of the gold was free. The losses above indicated appear enormous, but it must be remembered that the thrifty Chinamen throw nothing away—not even tailings; however completely, in their opinion, these may be exhausted, they still pile them up and keep them. When, for any reason, their mill would otherwise be idle, they re-pound and re-wash their old tailings, and always get some gold out of them. The piles of tailings are, however, left exposed, so that a considerable proportion gets washed down into the streams and rivers by the heavy rains that occur at each change of monsoon; and there are a good many Chinese of the poorer classes who make a sort of living by washing the sands in the river-beds, the gold they get being principally, to all appearance, that which has been thrown into the rivers by the miners up stream. It is noticeable that there is no gold, or very little, to be found in the rivers above the points where there are mines in operation. A fair day's work of one Chinaman in the river-bed (say six hours' actual work) was found, as the average of several trials, to produce an output of 7.3 grains of gold about .940 fine, worth say $0.28 U. S. currency. This quantity of gold was obtained by washing 22 large *dulangs* of gravel, each holding about 70 pounds of dirt.

It is interesting to note that in custom-milling, of which there is a good deal done here (many of the "fossickers" sending all the gold quartz they collect, whether by mining or picking out of the river-gravels, to one of the water-mills for crushing), the charge made is equal to $3.10 U. S. currency per (long) ton of quartz, this payment including the washing of the gold, but not, so far as I can make out, its cleaning and melting.

It is obvious from the above description, that the total quantity of stone crushed by all the mills in the district, supposing them all to be going simultaneously, and including the foot-mills, could not exceed some 12 tons a day at the best, an amount that could be far

more economically and efficiently handled in a five-stamp Californian mill of moderate power. Yet the total annual output of gold from this district (including, however, alluvial as well as reef-gold) is said to be 4000 taels (4861 ounces), fully .900 fine. The total number of men engaged in mining, in one way or another, is close upon one thousand.

There is, of course, nothing for us to learn from this imperfect and rudimentary gold-extraction process, which is doubtless destined to disappear ere long, before the progress of scientific mining, now making itself slowly felt throughout the far East. I think it advisable, however, to put on record all such crude efforts, if only to enable us to trace more completely the evolution of our modern systems of mining, and to teach us by what widely-divergent methods different races of mankind have attempted to solve one, apparently simple, problem.

THE MANUFACTURE OF LIQUID SULPHUROUS ACID IN UPPER SILESIA.

BY KARL EILERS, BERLIN, GERMANY.

(Glen Summit Meeting, October, 1891.)

THIS process, invented by Messrs. Hänisch and Schröder, of Rosdzin, was patented in 1883. It has attracted considerable attention in Germany, as a very neat and simple method of extracting sulphurous acid from the gases produced by roasting, and of condensing this acid so as to obtain it in the very compact anhydrous liquid form.

The special importance which such a process may have at this time in the United States is suggested by the growing inconvenience, amounting to distress, experienced in such localities as Butte, Mont., where the sulphurous gases from large roast-plants have proved not only fatal to vegetation but injurious to human health. It is reported that the authorities of Butte are earnestly considering the enterprise of removing, if practicable, the dense and suffocating cloud which now hangs over that city, and of which sulphurous acid undoubtedly forms the greater and most oppressive, if not the most poisonous, part.

May it not be that the solution of this problem is furnished by a

process which not only removes the sulphurous acid from the escaping gases of the roasting-furnaces, but also delivers it in a chemically pure condition and compact merchantable form? Whether it could, in this form, be transported at a profit to the present localities of those industries which require it, is a matter which need not here be investigated. I will only say, in passing, that I have no doubt there would be ways found of utilizing this important chemical if its production, even without profit, were shown to be the most feasible escape for great communities from a great nuisance.

This process rests upon the following well-established chemical and physical facts:

1. Sulphurous acid is absorbed or dissolved by cold water; and, practically, no other element of ordinary roast-gases is so absorbed.

2. Like other gases thus held in water (for instance, carbonic acid), it may be disengaged or expelled by heat and by agitation. The extreme form of agitation, which consists in atomizing the water, and the degree of heat which vaporizes the water, effect an almost complete separation.

To make these reactions economically available, it is obviously requisite that the sulphurous gases should be first cooled, in order to expose them to the action of cold water; that the watery solution thus formed should then be heated, so as to expel the pure sulphurous acid; and that the latter, escaping in the form of gas, should be liquefied by cooling and pressure for convenient transportation.

The process as practiced in Upper Silesia (in connection with the roasting of zinc-blende) presents, as will be seen, other features of detail, the purpose of which will be obvious. I need only observe, with regard to one of these, namely, the use of muffles in roasting, so as to obtain the sulphurous roast-gases separate from the combustion-gases of the fuel, that I do not suppose it would be absolutely necessary in cases in which muffle-roasting would not otherwise be deemed economically practicable. Its purpose evidently is to diminish the volume of the gases to be treated, and thus the size of the necessary machinery, the losses of heat, etc.

Other details, which appear to be universally applicable and advantageous, relate to the economy of heat, and may be so perfected as to make the initial heat of the gases suffice for nearly all requirements of the process. Of course this economy is highly desirable, and the devices by which it is effected are among the most meritorious original features of the invention.

Although I have seen this process in operation, I have been guided mainly, in the following description of it, by published state-

ments; and the accompanying plate is copied from the drawing filed by the inventors with the patents.*

As already observed, it is desired that the volume of the gases from which the SO_2 is to be extracted should be as small as possible; and hence muffle-furnaces are used for roasting. The type which appears to be most favored is the Eichhorn-Liebig furnace,† in which the fuel is gasified in a generator, and the fuel-gases are burned with pre-heated air in channels around the muffles in which the ore is roasted. These muffles are disposed to the number (in the original design) of six layers or stories over one another. The ore, ground very fine, is first dried on the top of the furnace, and is then introduced through a hopper to the upper muffle. In this muffle it is spread out, and is raked over occasionally during the 6 to 8 hours which elapse before the new charge is introduced.

After this period the old charges are all moved forward to make room for the new, and each of them (except the bottom one) drops through openings made for the purpose to the next lower muffle. From the lowest muffle the perfectly roasted ore is removed, after having remained in the furnace, in all, some 36 to 48 hours.

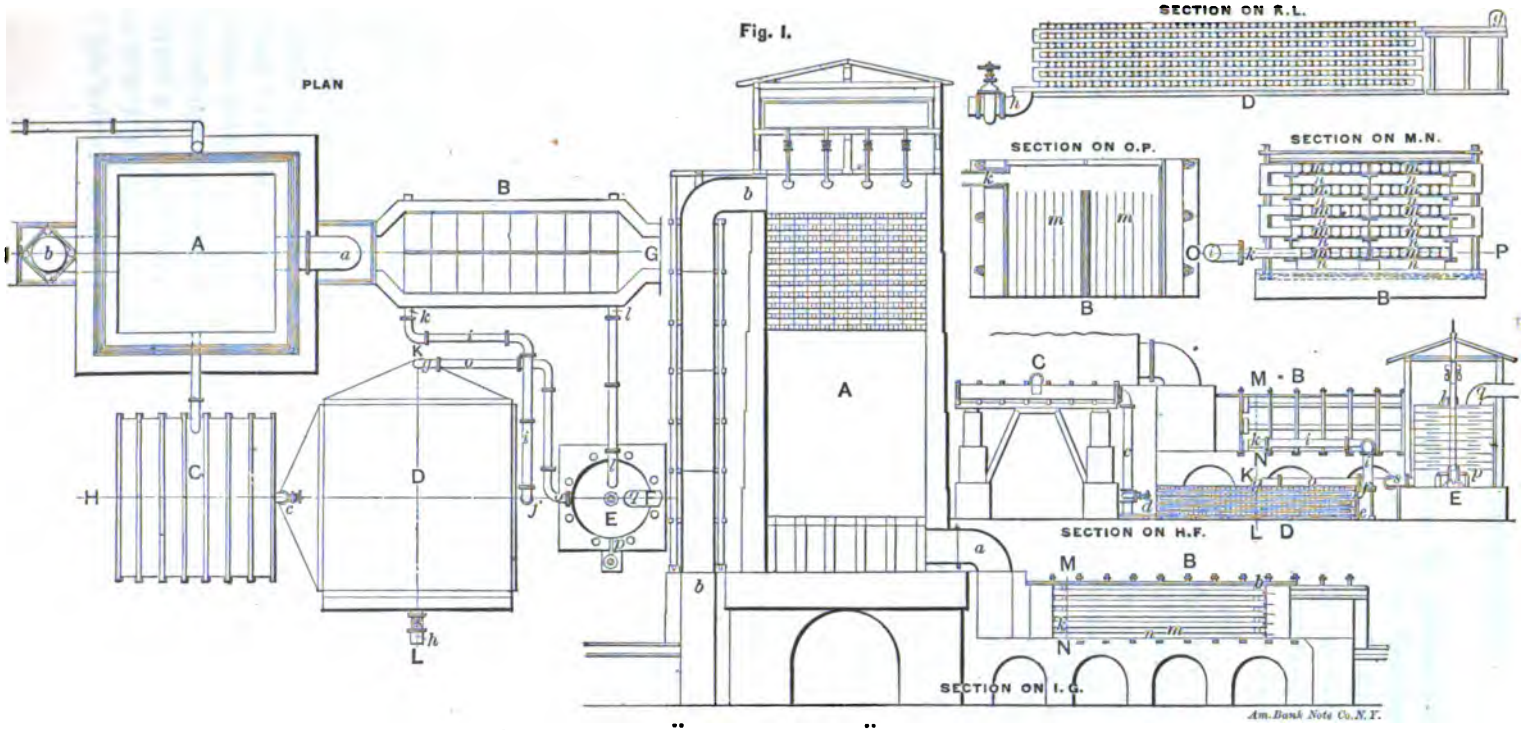
In practice it has been found that instead of the six-high muffle furnaces only three- or four-high muffles were necessary. This may be easily inferred from the following table, which shows to what stage of perfection the roasting had arrived on each of the four muffles of such a four-muffle furnace.

No. I., II., III. were three different zinc-ores such as are usual in Upper Silesia, and their contents of sulphur were as follows:

	I.	II.	III.
Raw zinc-blende.....	19.2	26.8	26.5
End of muffle 1.....	17.6	19.10 – 21.90	15.4 – 21.3
End of muffle 2.....	12.0	11.20 – 14.30	9.9 – 12.4
End of muffle 3.....	3.4	1.02 – 1.48	0.75 – 1.06
End of muffle 4.....	0.6	0.35 – 1.02	0.75 – 1.06

* Imperial German Patent, Class 40, Nos. 26,181; 27,581; 27,608. For descriptions in technical literature, see Kerpely's *Eisenhüttentechnik*, 1884–85, p. 386; also the volume for 1886, p. 328; and Bernhard Kosmann's *Oberschlesien, sein Land und seine Industrie*, p. 202.

† See German Imperial Patent, No. 21,032, Class 40, "Metallurgy;" also a description in the *Zeitschr. d. V. deutscher Ingen.*, 1886, p. 84, and an abstract of the same in Kerpely's *Eisenhüttentechnik* for 1886, vol. iii., p. 326.



APPARATUS OF HÄNISCH AND SCHRÖDER, UPPER SILESIA, FOR EXTRACTING SULPHUROUS ACID FROM ROAST-GASES.

The product of such a furnace amounts to 3000 or 3500 kilogrammes (3 to 3½ tons) per 24 hours, at an expense of 980 kilogrammes of coal (12 to 16 per cent. ash), and with an attendance of two men per 12-hour shift.

To produce ore that is roasted as well as the above, it is however necessary that the air entering the lowest muffle should be preheated. This is done by making the air for the roasting pass previously through channels provided for the purpose in the furnace itself, before it enters the lowest muffle. From here it passes up through the whole series of muffles, coming in contact with ore richer and richer in sulphur as it rises, and finally goes into the flue-dust chamber, where it is found to contain about 5 or 6 per cent. by volume of SO_2.

These gases are now to be freed from their sulphurous acid; but since at this point they have a temperature of some 400° C., they must first be cooled considerably for the double purpose of reducing their volume and of allowing the water to absorb and hold the SO_2 more easily.

This is done in the apparatus B shown in Fig. 1, which exhibits also the plan and various sections of the other apparatus necessary for the remainder of the process.

From B the gases pass through the pipe *a* to the tower A which is of considerable height (up to 35 meters), and is filled with coke or checker-work. They pass up through this tower, and are drawn off through the pipe *b* by a ventilator.

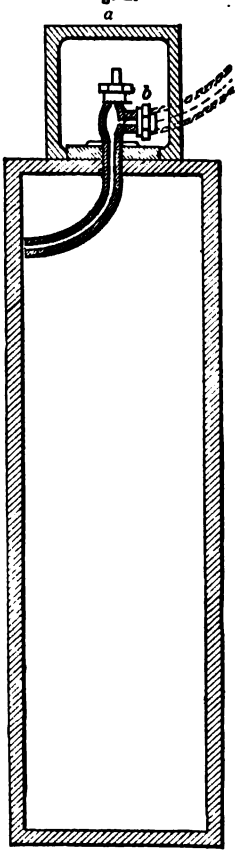
Before leaving this tower, however, they are freed from the sulphurous acid by water which comes from sprinklers at the top and trickles down through the coke, finally collecting in the reservoir C.

This is said to absorb the SO_2 entirely, leaving absolutely free from it the gases which pass out through *b*.

The tower and the reservoir, C, are placed together at an elevation sufficient to make the acid-water flow of itself through the whole of the remaining apparatus. In so doing, it flows first into D. This consists of a series of sheets of lead, laid one above the other, and separated from each other by inserting between each two sheets of lead bars of the same material. The ends of the sheets of lead are then connected with each other in such a way as to permit continuous passage for the acid-water from the coke-tower through the alternate spaces 1, 3, 5, 7, 9, etc., while other water from *e*, at a temperature of 100° C. flows through the intervening spaces 2, 4, 6, 8, etc., in a direction contrary to that of the water from C. In this way the

water from *e* at its hottest comes in contact with acid-water just about to leave D, and so already quite warm; and the water from *e* that is just about to leave D comes in contact with the new cold acid-water from C. Inasmuch as in this apparatus these two liquids are separated only by thin sheets of metal, it is possible here to give to the acid-water a temperature of about 70° or 80° C. This is, however, not sufficient, as it is necessary that 100° be attained, and so from D the water flows back to B, in which, as was previously stated the roaster-gases are being cooled.

Fig. 2.



CYLINDER TO HOLD LIQUID SULPHUROUS ACID.

B is arranged on much the same principle as D, except that the spaces between the sheets of lead are here larger, and that the lead plates at the bottom of each water-space are supported by a cast-iron plate.

In this apparatus it is the roast-gases that give up their heat to the acid-water, and this, when t has attained a temperature of 100° C. is drawn off through the pipe *l* to the tower E,

This is low, circular in shape, and provided at the center with a vertical lead-covered iron shaft, on which is fastened a series of horizontal disks that project towards the walls of the tower; while, on the other hand, to the walls of the tower are fastened horizontal rings which extend toward the middle of the tower, and project into the spaces between the disks on the shaft.

The acid-water on entering flows upon the uppermost of the disks, which, in revolving, flings the water to the wall. The first ring collects it again and delivers it back upon the second disk, which also throws it against the wall. The second ring feeds the third disk, and so on to the bottom.

In this way the water is changed into a sort of spray, and a large part of its SO_2 is thus disengaged. To remove the whole, superheated steam is introduced at the bottom of E, while the water is descending; and this, passing upwards, draws with it all the sulphurous acid, and some steam from the water. The latter water arrives at the bottom of the tower perfectly free from acid, but still

at a temperature of 100° C.; and in order that this heat may not be lost, the water is used in D, as before stated, to heat up fresh acid-water from C.

The steam and the sulphurous acid gas pass out from the top of the tower through *q* and enter a condenser. Here the steam is condensed and flows back to *e*, to be there freed from the relatively small amount of SO_2 which it may have taken up, or which may have been entangled with it in condensation. The sulphurous acid that was not so caught passes on through a dryer (at Hamborn a tower containing chloride of lime) and is there drawn into a compressor, where it is subjected to a pressure of five atmospheres. From here it is forced through a worm, which is surrounded by cold water to remove the heat of compression, and comes out at the further end in liquid form.

The compressor is very much like a two-cylinder steam-engine, the most important point of difference being that here the cylinder for compression is lined with phosphor-bronze. The reservoir into which the liquid acid is forced, is however only wrought-iron.

To be sent into commerce, the acid is transferred to cylindrical vessels of wrought-iron of the form shown in Fig. 2. They have a stopcock at the top *b*, protected with a cap *a*.

If it is desired by the consumer to use the acid in the form of gas, the cylinder is set with its axis vertical, the cap *a* is removed and the stopcock is opened. The gas will then flow till the temperature inside the cylinder has fallen to — 10° C., when the gas will no longer be generated till the temperature has risen again.

To obtain the acid as a liquid, it is necessary to lay the cylinder so that its axis is horizontal, and the spout *b* is directed upward. A tube is then fastened to *b* and the stopcock is opened. As the pipe on the inside is bent close against the wall of the cylinder, it is possible to obtain nearly the whole of the sulphurous acid in the liquid form.

The pressure on the inside of these cylinders is 0 at —10° C., but at 40° C. (the temperature out in the sun in summer) it amounts to 5.15 atmospheres.

TANDEM TANKS FOR HOISTING WATER FROM FLOODED SLOPES.

BY J. H. BOWDEN, WILKES-BARRE, PA.

(Glen Summit Meeting, October, 1891.)

THE water-hoisting tanks herewith illustrated have been designed for removing large quantities of water from recently-flooded mines, through their hoisting-slopes, with rapidity proportional to the capabilities of the hoisting machinery available, the tanks being adaptable to slopes of small sectional areas and varying pitches.

The following features may be of interest to those operating mines liable to be flooded.

1. The arrangement of doors on each tank by which it may be automatically filled by immersion at any point on the varying pitch of a slope, the water being retained while hoisting on the flat as well as on the steep pitches, and quickly and automatically discharged at the top.

2. The arrangement, in connection with the above-mentioned doors, of side-wheels over the rear wheels of each tank and of side-dumping tracks at the top of the hoist, for the automatic emptying and quick return of the tank.

3. The arrangement of two or more tanks, one in front of the other, so as to give as large a capacity in a single hoist as the engine-power will permit, yet without making any tank too large for the sectional area of the slope, or of unwieldly length, or of such a shape that it cannot easily pass over vertical curves; and without concentrating too much weight on any one pair of wheels or on any point of the track.

The details of construction and method of operating are shown by the accompanying sketches (Figs. 1 and 2), and the following more detailed description.

At the end of each tank is a large iron door of almost the full size of the end of the tank, opening inwards, so that when immersed the tanks fill almost instantly. To provide for holding the water while it is hoisted up flat pitches, a wooden door is attached to the front of each tank, opening outward. Each front door is attached to the

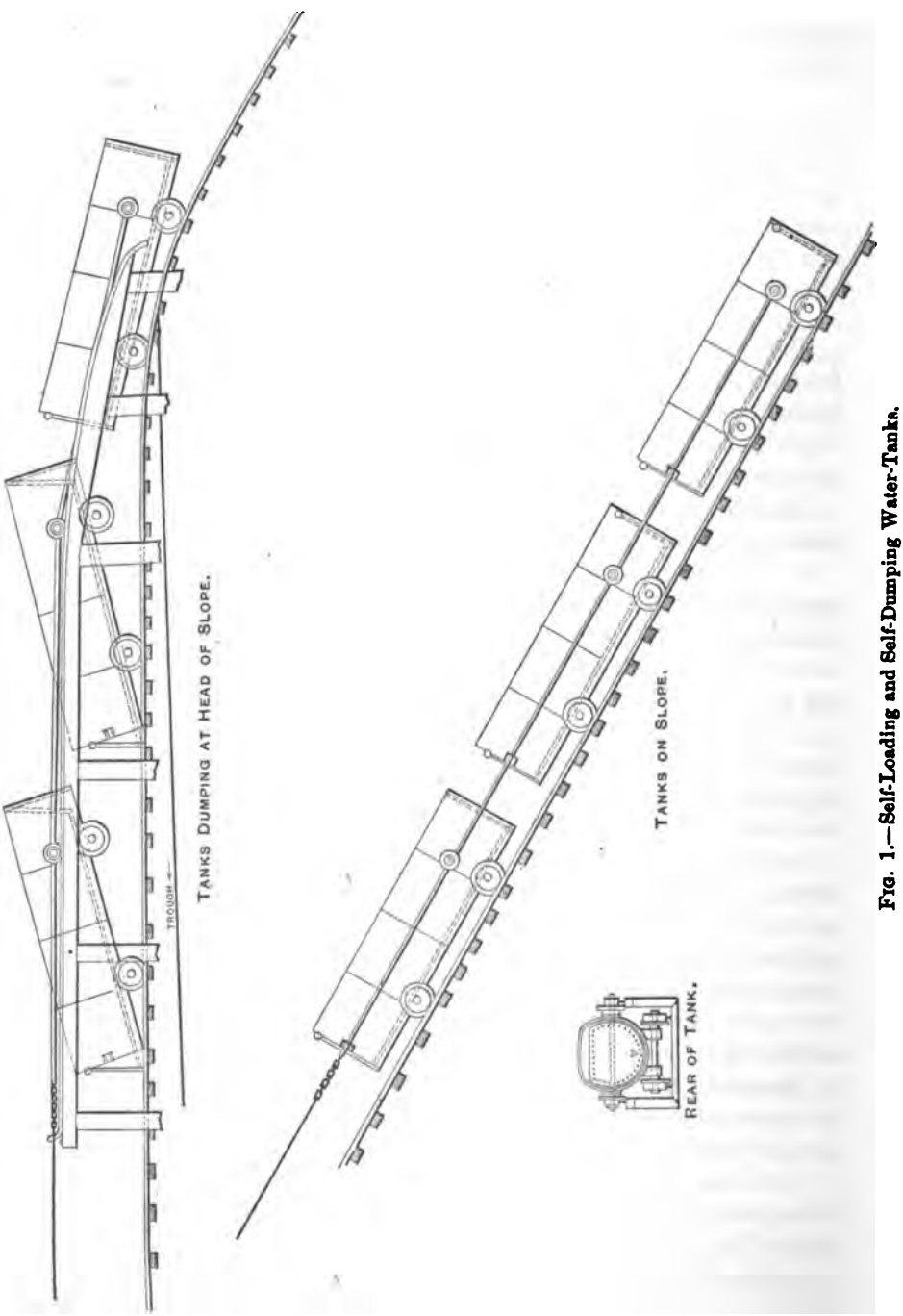
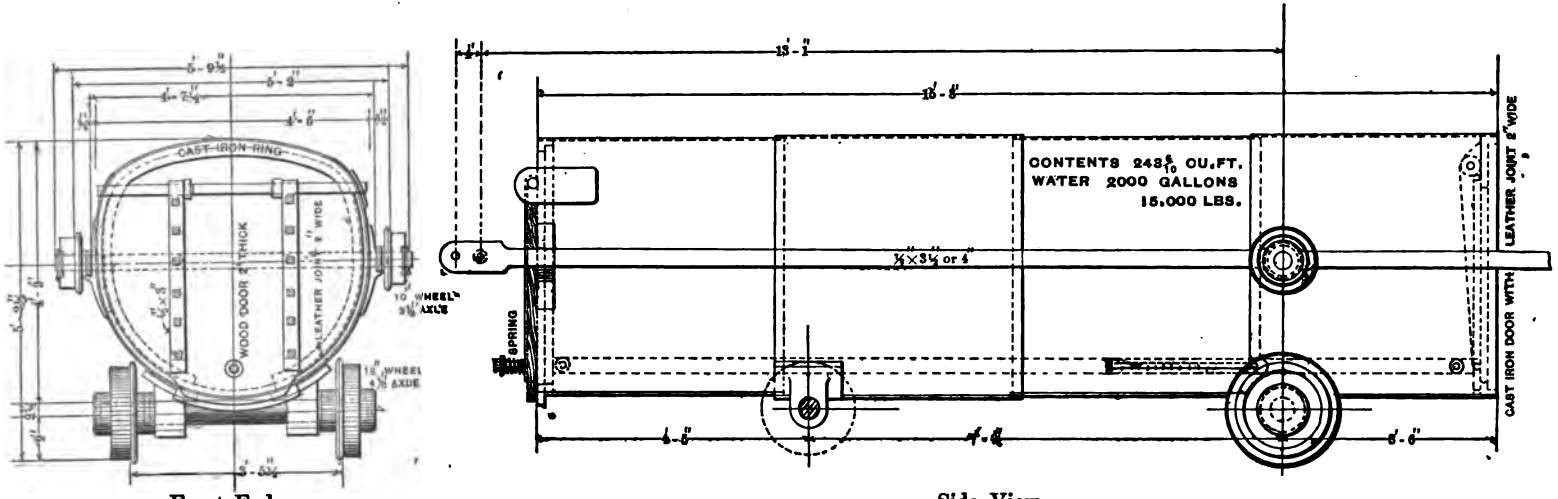


FIG. 1.—Self-Loading and Self-Dumping Water-Tanks.

FIG. 2.



Front End.

Side View.

Self-Loading and Self-Dumping Water-Tanks.

door at the back by an iron rod, provided with a sliding link, so that the back door can open independently of the front; but the latter is held closed as long as the rear door is closed. This connecting-rod, as shown in Fig. 2, passes through the front door and through a spiral spring in front of it, so that the amount of pressure necessary to keep the water from leaking out may be readily applied.

The tanks are mounted on self-oiling closed wheels, so arranged as to exclude water from the bearings while the tanks are immersed, and to retain the lubricant.

Each tank is provided also with side-wheels, vertically over the rear axle, which have a gauge sufficiently wide to clear all other portions of the tank; and on the surface an elevated track is provided, upon which these dumping-wheels run and thus raise the rear end of each tank as much as may be necessary to dump the water into a trough between the tracks; the tilting forward of the tanks opening the back door and releasing the front one. The tanks while emptying rest on their forward wheels and on the dumping-wheels. By having the tracks at the surface slightly up-grade, the tanks will run back when empty, as soon as the rope is slackened. To allow dumping, the hoisting-rope is attached to the tanks by a yoke reaching back on the sides and pivoting on the axle of the dumping-wheels, the tanks back of the first one being attached by eye-bars reaching from axle to axle of the dumping-wheels on the tanks. A stop is provided, to prevent the yoke on the forward tank from dropping and catching in the track when the rope is slackened.

This plan of "tandem tanks" was designed and used to hoist about 25,000,000 gallons of water which had been admitted to extinguish a mine-fire in one of the Susquehanna Coal Company's mines. The slope was small in section, and 3200 feet long, with single track, and with pitches varying from 4 to 20 degrees. The hoisting-plant consisted of a pair of 26 inches by 60 inches direct-acting engines with cast coned drum, 9 to 12 feet in diameter, carrying 1⅛-inch steel rope. These engines had been previously hoisting five cars, weighing about 4 tons each when loaded.

APPARATUS FOR THE MANIPULATION OF IRON AND STEEL PLATES DURING THE PROCESS OF FINISHING.

BY GRAM CURTIS, PITTSBURGH, PA.

(Glen Summit Meeting, October, 1891.)

THE apparatus described below for the handling of plates upon their delivery from the rolls, embodies features believed to be of interest to those contemplating the building of new plate-mills, and also to such owners of existing mills as may be considering the improvement of their works.

The points to which attention is called are the small space required for the finishing-end of the mill, and consequently the very moderate size of the building for the amount of plates handled; the directness and ease of manipulation, and therefore the small amount of labor required; the distribution of plates; the facilities and time allowed for their cooling. These, together with other features of detail and general arrangement scarcely less important, will become apparent when the descriptions and illustrations herewith given are understood.

As these devices deal with the finishing of the plates only, they may be added to any plate-train when the floor-space and the building are large enough to permit their introduction; and it will be obvious that the features of this method, more or less modified, are applicable to any existing mill. Figs. 1 and 2 show a three-high 32- and 19- by 120-inch Lauth mill, with hydraulic lifting-tables, served by four heating-furnaces, with hydraulic cranes for charging and drawing.

Upon leaving the rolls, the plate is allowed to rest with its forward edge projecting somewhat beyond the end of the catcher's table. Here it is seized by the automatic tongs, *A*, suspended from a buggy travelling in the overhead trough, *B*, and actuated by the hydraulic cylinder, *C*. These tongs are operated by a boy, who stands several feet from the heated plate, and controls the long light handle, *D*, shown in the plan, Fig. 1. His duty is merely to advance the tongs by means of the hydraulic cylinder, *C*, to raise or

lower them by the handle, so that their jaws come on a level with the edge of the plate, and, when they have taken hold, to reverse the valve, and thereby draw the plate over the straightening-table. The construction of the tongs is such that they open by advancing upon the plate and close upon being withdrawn. The stronger the pull the tighter their hold. They have a vertical motion of over 24 inches, and are balanced so that their elevation or depression involves no effort. They are also readily reversed, so that the plate may be seized by the other end and drawn towards the rolls should occasion require it. When the plate has reached its proper position on the straightening-table it is stopped; the tongs are caused to advance upon the plate, whereby their grip is released, and they are easily thrown off by means of the long handle.

The straightening-table, some 63 feet in length, is composed of heavy-section railroad-bars, placed about 12 inches apart and resting on heavy castings, which are bolted down upon solid foundations. The inner ends of these rails are firmly secured to the castings, while their outer ends, though held in place, are free to expand, and the supporting castings are placed so wide apart that they receive but little of the heat radiated from the plate. This manner of carrying the plate permits a free circulation of air all around it. The cooling-process may be still further favored, and much radiation from the floor beneath may be cut off by covering this floor with a few inches of water, for which purpose it is depressed and paved.

From the arrangement and position of the drag-out, just explained, in relation to the straightening-table, it is evident that plates even up to 63 feet in length can be handled with the greatest ease. The boy conveys the plate to its proper position on the table, and here it lies, after receiving the straightening required, until its place is needed for a succeeding plate. It is then transferred laterally to the cooling-bed. This bed, formed by the extension of certain rails of the straightening-table, is composed of three separate elementary beds. The transfer of the plate is effected by endless chains,—two to each bed,—furnished with heavy conveying-blocks. These blocks ride upon the rails and carry the plate with them. The spaces or divisions of the chain are, of course, sufficient to receive the widest plates that are rolled on the train. By a simple arrangement of levers and clutches (not easily shown on so small a scale), the three beds, at a moment's notice, can be made to work as one large bed, or the middle bed may be split in two, each half acting with the adjacent elementary bed, thus forming *two* beds of equal size. Or the middle bed

may be made to act in unison with either one of the side beds, forming two beds, but of unequal size. These combinations, it will be readily seen, adapt the cooling-bed to the different lengths of plates. For instance, in rolling plates of 20 feet or under, the three beds are used; for plates from 20 to 30 feet in length, two beds; and for plates from 40 to 63 feet, one bed would be used. When a plate 30 to 40 feet in length is rolled in a run of smaller plates, it is placed on the table so as to cover the middle bed and one of the side beds, and these two beds are then worked in unison until this plate is delivered. The straightening-table and cooling-beds will accommodate fifteen 20-foot plates; hence, if we require five minutes for the rolling of a plate of this length, we will have one hour and a quarter as the time which may be allowed for cooling. This time, of course, would be diminished in handling longer plates. But the cooling could be gently hastened by the introduction of a series of air-blasts, from a fan, beneath the plates, as they lie upon the bed.

As space on the table becomes needed for a new plate, those undergoing cooling upon one or more of the beds—according to the length of the new plate—are advanced one division of the chain, thus clearing that part of the table and furnishing the room required.

These beds require the services of but two men, who, by means of the levers referred to above, arrange the clutches actuating the chain-wheels, *Q*, and operate the hydraulic cylinder, *E*, under the floor, which furnishes the power for moving the plates. The same two men, by an arrangement about to be described, also turn the plate over and trim its two longer edges.

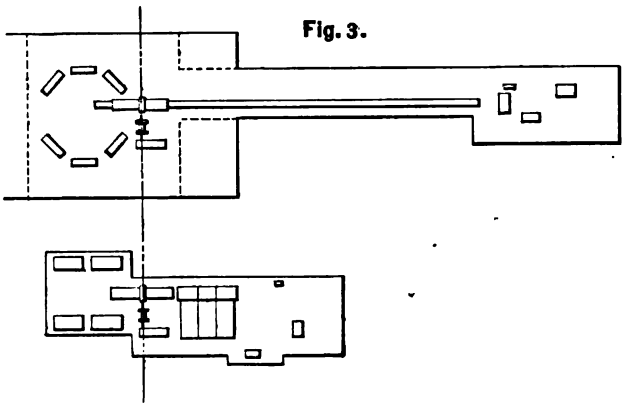
The cooling-bed is very low, being 17 inches above the floor at the straightening-table, and rising at its delivery-end to 30 inches,—a very convenient height for the laying-out of plates. But the manner in which they are here handled and carried before the slitting-shear does away with much of the laying-out that is ordinarily required.

On its delivery from the cooling-bed (which is accomplished by merely advancing the chains), the plate passes upon the buggies, *F, F*. These buggies receive the plate, each on a single line of rollers; and to facilitate its motion away from the bed, these rollers have a pitch of about 2 inches to 4 feet. When the plate is well upon the buggies, the men in charge step between it and the bed. One at each buggy sets the gauge which determines the width to which the plate shall be sheared, and presses a handle, which releases a catch, and permits the inner end of the line of rollers to drop until the incline is

2 inches to 4 feet in the other direction. The plate, thus assisted by gravity, is then rolled against the gauges by a very slight effort on the part of the men. In this position it is securely clamped, and by means of another underground hydraulic cylinder, *G*, and a wire-rope upon which the buggies have a movable grip, the plate is fed to the slitting-shear, *H*, and is there cut by one of these men. The movable grip permits the buggies to be readily disengaged from the rope, and placed to suit the length of the plate to be handled. A carrier or temporary buggy is set between the regular buggies when an extremely long plate is to be sheared.

As the buggies travel upon a V-shaped rail, the plate is run before the shear with great exactness, and a true cut is the result. After the cut has been made, the plate is brought back to the entering side of the shear. Here, by means of the suspended tongs, *I I*, and the

Fig. 3.



hydraulic cylinder, *J*, it is turned over, and, after proper adjustment against the stops, it is clamped, and again run before the shear.

As all the heavy motions are performed by power or by the aid of gravity, the two men referred to can handle with ease plates weighing 3 tons or even more, furnishing all the labor necessary to pass them through the slitting-shear, trimming their long edges, and deliver them upon the castor-rollers for the guillotine-shear, *K*. Here another gang takes them, cuts them to the lengths required on this shear, or shapes them by the trimming-shear, *L*.

The cranes, *M* and *N*, are for the convenient handling of heavy plates, for storing, etc. *O* is a crane reaching outside of the building for shipping, and *P* is the scrap-shear.

The comparative size of the Homestead plate-mill of Messrs. Carnegie, Phipps & Co., and the mill designed for about the same capacity on the plan here described, is shown in Fig. 3.

As a summary of the principal advantages claimed in the arrangement thus described, I may mention:

1. The moderate first cost (considering the output) arising from the small ground-space and the small building necessary, and from the cheap though durable nature of the machinery employed.

2. The economy of operation, due to the convenience of handling and the small amount of labor required.

3. The ease with which many, if not all the devices described, may be applied to existing mills.

In this connection it should be added that, although hydraulic power has been shown in the illustrations here given, the whole apparatus has also been worked out so as to be operated by a steam-engine.

THE HANDLING OF INGOTS AND MOULDS IN BESSEMER STEEL-WORKS.

BY GRAM CURTIS, PITTSBURGH, PA.

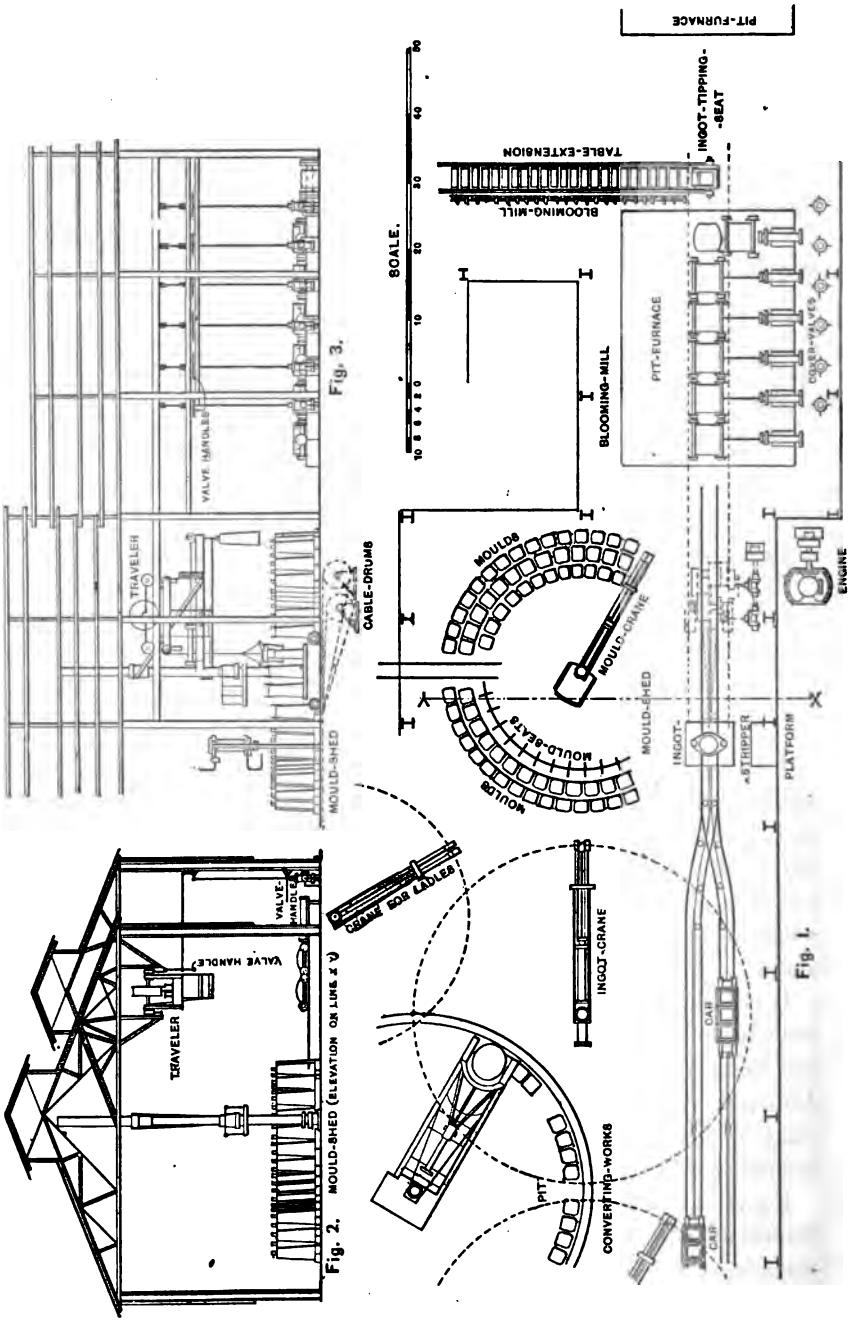
(Glen Summit Meeting, October, 1891.)

THE keen and close competition now ruling in the iron and steel manufacture requires imperatively, in the design and construction of the machinery employed, the fulfilment in the highest practicable degree of two cardinal conditions: economy of labor in operation, and durability. The daily pay-roll and the cost of repairs far outweigh in importance the interest on the first cost of this part of the plant.

It is believed that the apparatus here described will be found to embody, in these respects, a substantial improvement upon the present manner of handling ingots and moulds at our large steel-works. Although it includes a number of novel features, it consists essentially in the choice and grouping of certain approved economical devices specially suited to the particular service required.

Fig. 1 is a plan, and Figs. 2 and 3 are elevations of parts of a Bessemer converting-house, together with a small portion of the blooming-mill—sufficient to illustrate clearly the means and manner of handling the ingots and moulds.

A short cable-road of simple form, somewhat similar to those used



Apparatus for Handling Bessemer Ingots and Moulds.

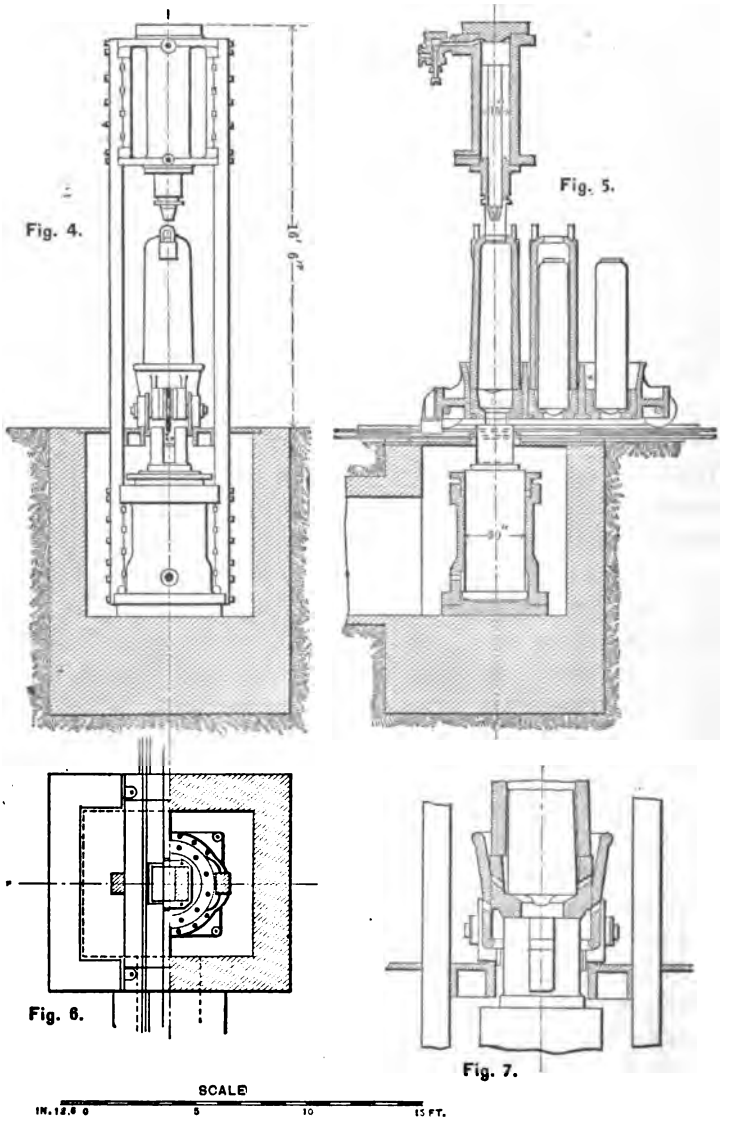
in nearly all large cities, is laid on the floor of the converting-house, within the orbit of the ingot-cranes. The two tracks of this road converge into one just before reaching the ingot-extractor. The movements upon these tracks of the cars carrying the moulds and their enclosed ingots, and also the action of the stripper, which forces the ingot free, are under the perfect control of one operator, stationed on an elevated platform adjacent to the last-named machine. After passing through the stripper the car is halted in the position shown in Fig. 3, where the (now loosened) hot mould is lifted off and conveyed to a cooling-seat by the mould-crane, which, on its return-trip to the car, picks up a cold mould, replacing therewith the one just removed.

The mould-crane is operated by a boy, standing on the platform which it carries, and from this point not only the hooks, which engage the mould, are handled, but all the motions—the rack, the lift, and the swing—of the crane are controlled.

While the removal of the hot moulds is in progress the traveller, or overhead buggy, is engaged in conveying the stripped ingots to the furnace and placing them in the pits; also in drawing the heated ingots and placing them on the tipping-seat, whence they are delivered to the blooming-train. This buggy is, in fact, a light, quick-working, travelling crane; it is actuated by wire rope, and is capable of all the motions necessary for thorough handling. Besides the lift, longitudinal travel, and a side-movement by which it completely covers the furnace-pits, it has power to seize or release the ingot and also to revolve the tongs through an arc of 180°. The whole is under the control of a man seated upon a platform suspended from the buggy-frame; and, by bringing the handles to the furnace-cover valves within his reach—through a simple bell-crank and a couple of rods—the same man is enabled to open and close the furnace-pits during his operations of charging and drawing. In order to enable him also to manipulate the covers while the car is yet in motion, approaching or receding from its position directly over the pit, the valve-handles are made horizontal bars, some 6 feet long, making the valve accessible during that length of travel. This arrangement utilizes the time expended in the stopping and starting of the buggy.

As the ingot-extractor (Figs. 4, 5, 6, and 7) forms a sort of center or nucleus of the system, and also possesses some novel characteristics, it deserves a description, somewhat in detail, of its construction and operation. It is of the vertical type, and consists of two hydraulic cylinders connected by heavy steel bars, as shown in Fig. 4. The

ram of the smaller cylinder, placed above, works downward, and has a stroke of 3 feet 9 inches. The larger cylinder, 30 inches in



Ingot-Extractor.

diameter, with a stroke of 11 inches, is in a pit bridged by the rails and covered with floor-plates. The car from the casting-pit is stopped with the first ingot directly under the upper cylinder, the

ram of which is then caused to descend, by exhausting the water-pressure from below it, until it rests upon the ingot. During the descent water (not under pressure) flows in above through the balanced check-valve shown, and becomes imprisoned by that valve the instant the ram ceases to move. The lower cylinder—capable of lifting some 280,000 pounds—is now put in operation, and the movable seat which carries the mould, but not the ingot, is lifted, forcing the mould from the ingot, which quietly drops upon the seat, where it remains after the mould has been lifted off by the crane. This is more clearly shown in Fig. 7. In its trip back to the converting-house the car runs upon its proper track automatically, without any attention from the workmen.

In the mode of handling which I have described, much of the arduous labor ordinarily performed at the pit vanishes, and this point ceases to be a throttle upon the output of the plant. Much of the heat is carried to a point where it can be more easily dealt with, where the hot moulds and ingots can at least be handled from behind screens and at a distance.

By the plan of mould-shed shown, a whole storehouse of moulds may be conveniently kept within immediate reach, and this without the aid of a locomotive and mould-cars, thus saving the yard-room needed for the requisite tracks and switches. The large number of moulds at command permits time for slow cooling by the draft which their high temperature naturally creates, an item which, in combination with the powerful extractor, will contribute in no small degree to the material prolongation of their term of service. To facilitate the circulation of air in and around them and to permit their ready inspection and repair (by the cementing of cracks and defects), the moulds may be placed on open seats supported on posts, and arranged over a circular pit some 4 feet deep.

The simplicity and directness of all the motions given to the moulds and ingots during their manipulation; the attachment of the extractor, with its enormous lifting-force, to the regular pressure-system direct, without the intervention of an auxiliary pump; the promptness of its operation, and the small quantity of water it requires; the readiness with which the mould-cars may be detached from the cable and temporarily handled by the narrow-gauge yard-locomotive in case of accident to the drag-out rig, etc., are features of the apparatus to which particular attention is invited. Instances of long and successful service of the different devices here combined are too common to require special mention. I will therefore content myself with pre-

senting, as the best data for criticism and comparison with present practice, the statements tabulated below concerning the number and duties of the workmen required for the operation of the apparatus described.

Men Required for the Handling (i.e., the Setting of Moulds in Pit, Stripping, Heating, and Delivering to Blooming-Train) of 420 *Ingots (Equivalent to over* 840 *Tons of Steel) in Twenty-four Hours.*

1 man setting moulds in pit.
2 pulpit-boys—1 to each ingot-crane.
N. B.—These cranes lift, rack, and swing from the pulpit.
4 men to hook moulds at pit, guide them to seat in cars, etc.—2 men to each ingot-crane.
1 man at ingot-stripper, handling cars and stripping ingots.
1 man on mould-crane, lifting off hot moulds and replacing them with cold.
1 man on traveller, conveying ingots from car to furnaces; also from furnaces to blooming-train.
2 heaters—1 on each furnace.
2 heaters' helpers—1 on each furnace.
4 bottom-makers—2 on each furnace.
1 man at tipping-seat.
1 man at engine.

Making a total of 18 men and 2 boys per turn, and dispensing with the customary locomotive, its driver, and switchman.

This is believed to be considerably less labor than that required for the same purpose at any existing plant yielding the same output.

ELECTRIC LOCOMOTIVES IN GERMAN MINES.

BY KARL EILERS, BERLIN, GERMANY.

(Glen Summit Meeting, October, 1891.)

DURING my stay of two years in this country, I have heard of only four electric mine-locomotive plants, of which three were built by the firm of Siemens & Halske, and the fourth by the *Allgemeine Elektricitätsgesellschaft*, both of Berlin.

The earliest of these was established in 1882 in the royal coal-mines at Zaukeroda, Saxony. In 1883, the Consolidated Paulus and Hohenzollern colliery at Beuthen put in a plant; and in the

beginning of 1884 the plant at New Stassfurt was got into running order. These three are the installation of Siemens & Halske.

The *Allgemeine Elektricitätsgesellschaft* has built thus far only one plant for mining purposes, and that was solely an experimental one, established in the fall of 1890 at the coal-mine Richterschacht I. of the Vereinigte Königs- and Laura-hütte, in Upper Silesia.

Of the installation at the Hohenzollern mine, which will, perhaps, serve best as a type of those built by Siemens and Halske, I am able through the kindness of Inspector Kraski, in Beuthen, to present some sketches. At this mine there are at present two lines on which electric locomotives run. Each has its own locomotive, and each locomotive has had, till recently, its special dynamo above ground; but both dynamos were run by the same steam-engine.

This engine was not originally designed for the purpose of running dynamos, having served in its early days to operate a Cornish pump, in which work it developed 120 horse-power. Now it furnishes electricity to run the two electric locomotives (together 43 horse-power) and yields, besides, 36 horse-power to an endless chain, also run by electricity in the same mine, thus attaining to only about two-thirds of its normal power. Before the end of this year, however, the company expects to have arranged its locomotive-system and the endless chain so that the full power of the engine can be used. This it will accomplish by making the trains run in a continuous circuit, instead of running shuttle-trains, as is now done.

From this engine the power was transmitted to the dynamos by means of belting, so that the armatures made about 800 revolutions per minute.

As already observed, there were, till recently, two of these dynamos, each having about 23 horse-power and being of that type of Siemens & Halske known as D_o. They were compound-wound, and the armatures were on the principle of Hefner-Alteneck. The brushes were lamellæ of copper, which had to collect a current of 34 ampères at 320 volts.

These dynamos the company has now replaced with two others of a newer type ($_gH19$) of 60 horse-power each, and yielding 140 ampères at 300 volts. Only one of these is kept in use; the other being held in reserve.

At present, the current is transmitted through the shaft by two cables for each locomotive. These consist of a core of rubber

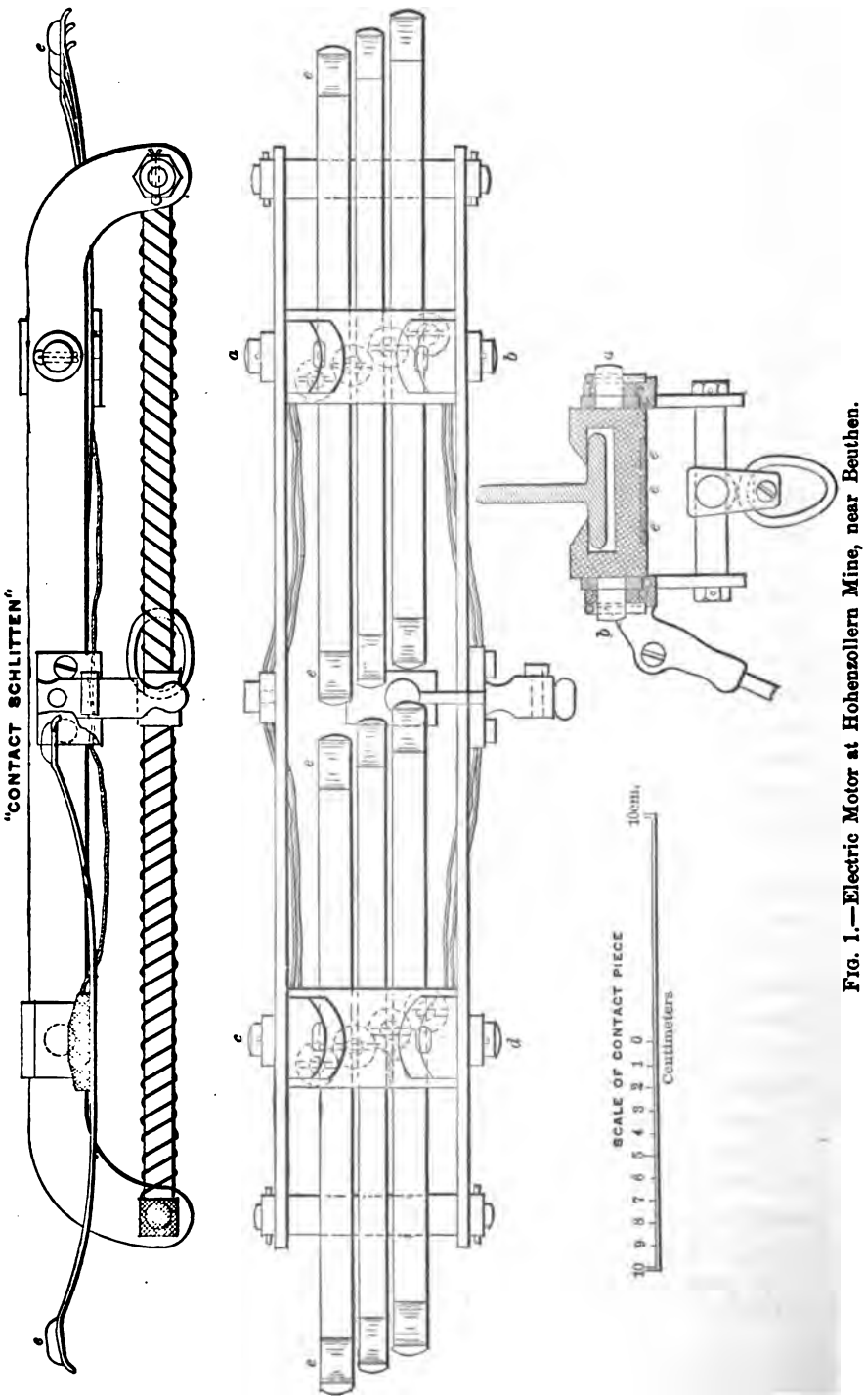


FIG. 1.—Electric Motor at Hohenzollern Mine, near Beuthen.

with 19 copper wires tinned and wound with impregnated hemp, the whole being protected by 13 pieces of galvanized iron wire, 2½ mm. thick. It is cable No. 4323 of Felten and Guilleaume, in Cologne.

This cable is fastened to the shaft-wall with iron hooks, immediate contact between cable and hook being prevented by surrounding the former at the point of support with a layer of wood or asbestos. In this way the loss of electricity is reduced to a minimum; whereas, formerly, when they used a different cable, with lead covering which was in immediate contact with the iron hook, great loss of current was caused by the destruction of the lead covering at the points of support, followed by destruction of the insulation also.

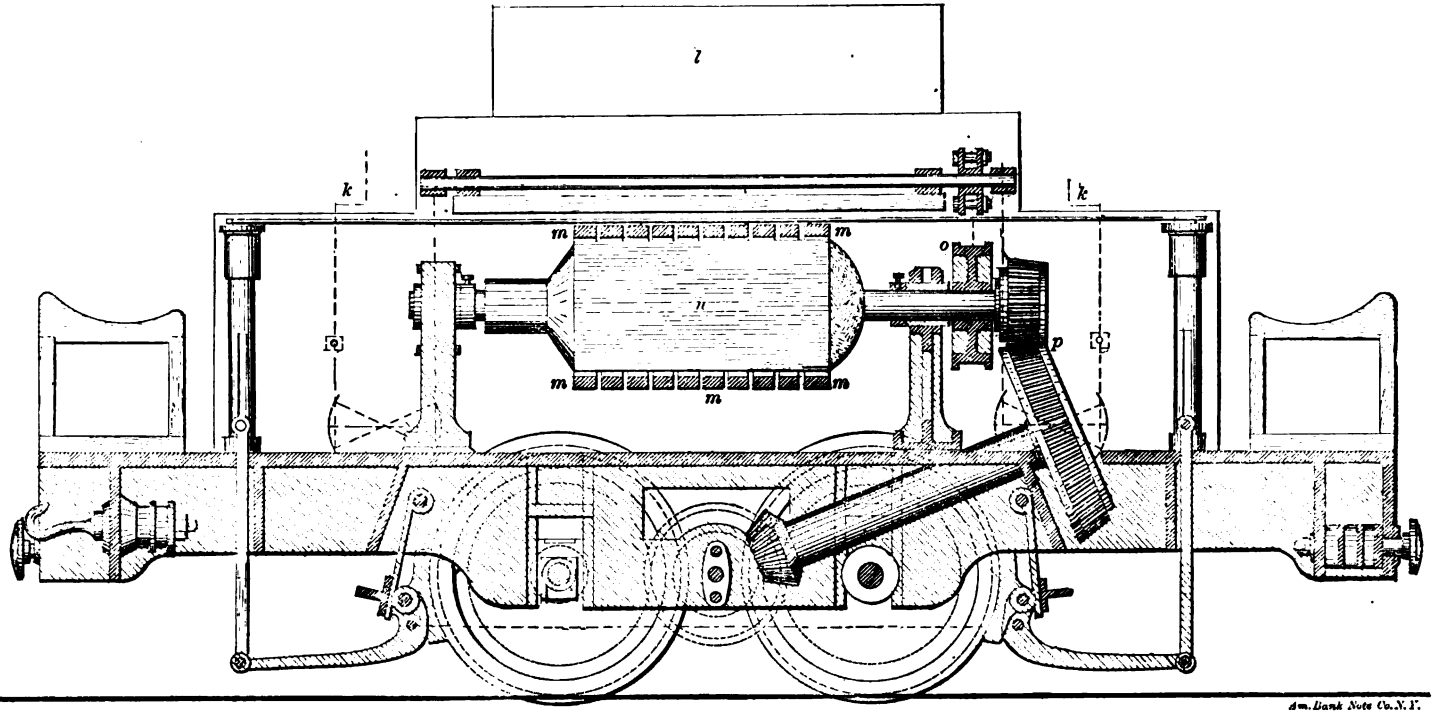
The cables, 230 meters long, carry the current to the ⊥-irons that serve as contact-rails. From these it is transmitted to the motor by a *Kontaktschlitten* (contact-sled), as the Germans call that piece of apparatus which is in sliding contact with the ⊥-rail, and connected with the locomotive by a short piece of cable. In the section of the *Kontaktschlitten*, Fig. 1, the ⊥-iron is shown, and in Fig. 3 the two contact-rails are shown with their supports above the locomotive.

The supports for the ⊥-irons are placed at intervals of four meters. They consist of tinned cast-iron frames, having on the lower side two cup-shaped openings. Into these openings project the iron bolts to which the ⊥-rails are immediately fastened; but the two pieces of metal (the cast-iron frame and the bolt) are separated from each other by a cup-shaped piece of rubber, inside and outside of which sulphur has been cast, so that the bolt is held rigidly in position, and is at the same time completely insulated from the cast-iron frame. The frames themselves are fastened to the roof of the gangway by iron bolts. A hole is drilled into the rock; into this is driven a wedge of wood, and into the wood is screwed directly the iron bolt. Or, if the gangway happens to be timbered at a convenient point, the iron bolt is fastened directly to the roof-timber.

The single lengths of ⊥-iron are fastened together with plates bolted and soldered to the ends to be joined.

The construction of the contact-sled is shown in Fig. 1. It consists of a frame of iron which supports the two brass castings at *a*, *b*, and *d*, *c*. These fit over the ⊥-iron (see the section on *a*, *b*) and the projecting pieces of the upper part are drawn against the top of the rail by six steel springs, *e*, which press against the under side of the ⊥-rail in such a way that through the contact of the brass above

FIG. 2.



Am. Bank Note Co. N. Y.

SCALE OF LOCOMOTIVE AND INSULATING SUPPORT.

10 5 0 10 20 30 40 50 100 150 200 250 Centimeters

ELECTRIC MOTOR

AT

HOHENZOLLERN MINE NEAR BEUTHEN.

and these springs below, the current is easily conducted from the rail.

When the "sled" was first introduced, an attempt was made to lessen its friction by providing on the frame oil-boxes, which were to keep the contact-rail well lubricated. It was found, however, that the oil oxidized very rapidly, and then the ⊥-rails soon became, contrary to all desires, excellently insulated. The contact-sleds are therefore now made without oiling arrangements.

From the middle of each of the sleds a hemp rope runs down to the locomotive. Its purpose is to drag the sled while serving also as a center about which the conducting cables are wound, loosely, so as to be relieved from all strain.

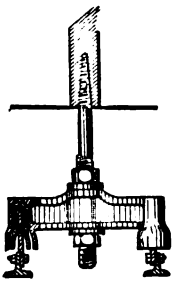
Of the locomotives actually in use, I have not been able to obtain drawings. But Inspector Kraski has kindly furnished me with a drawing (Figs. 2 and 3) of a locomotive, designed at the Hohenzollern mine after several years of experience with the original Siemens & Halske machine. The Berlin firm was not willing to construct a locomotive on this plan exactly, but sent new locomotives, which differ from this design only in having lower driving-wheels, and in not having a brake on the armature-shaft at *o*. In the drawing is shown, at each end, the seat for the driver, as is usual. Also the two crank-arms, by means of which the current is sent through a greater or less resistance in the box at *l*, before going around the magnets, *m*, and into the armature, *n*.

The two most important improvements of Inspector Kraski are at the points *o* and *p*. A great deal of difficulty was experienced from the fact that, notwithstanding the many resistances at *l*, the armature always started to revolve very suddenly when the current was turned on. The result was, that very frequently teeth were knocked out of the bevel-wheels at *p*. Hence, the larger wheel at *p* is now made of cast-steel, and fastened to the shaft somewhat elastically with springs, and the smaller wheel is of phosphor-bronze. By using these, the breaking out of teeth was remedied.

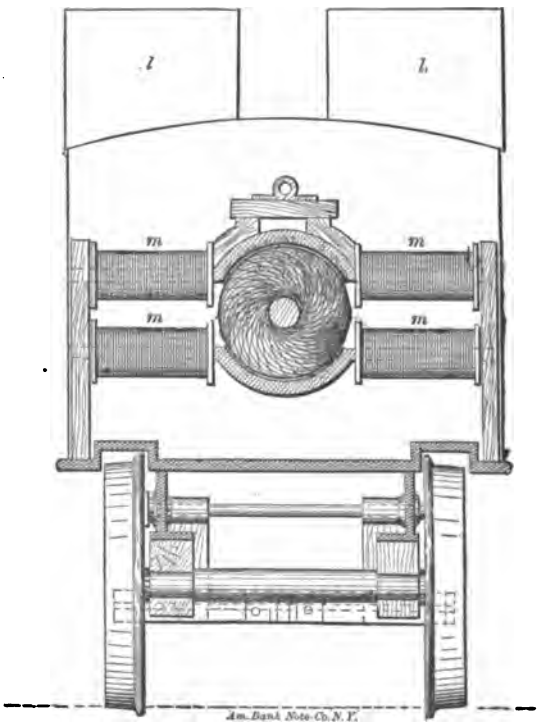
Immediately behind these, at *o*, another wheel has been fastened to the armature-shaft, on which a brake is to act, the object being to enable the driver to apply the brake as nearly as possible to the source of the power. This feature, as I have remarked, Siemens & Halske refused to adopt. It was not, by any means, the intention of the inspector, in designing it, to abandon the brakes applied on the driving-wheels, but only to use the additional brake in starting the locomotive.

As the drifts are generally wet, it has been found advisable to put a sand-box on the locomotive to prevent the slipping of the driving-wheels.

FIG. 3.



INSULATING SUPPORT
IN
RELATIVELY CORRECT SIZE & POSITION
ABOVE LOCOMOTIVE.



Cross-Section and Elevation of Fig. 2.

As far as I can learn, the arrangements at Zaukeroda and at New Stassfurt are not essentially different from those described here, ex-

cept that at Zaukeroda I am told the *Kontaktschlitten* are replaced by *Kontaktwagen*, the difference being that the latter run on wheels instead of simply sliding along. This principle was used at Zaukeroda from the beginning, and the engineers are very well satisfied with it.

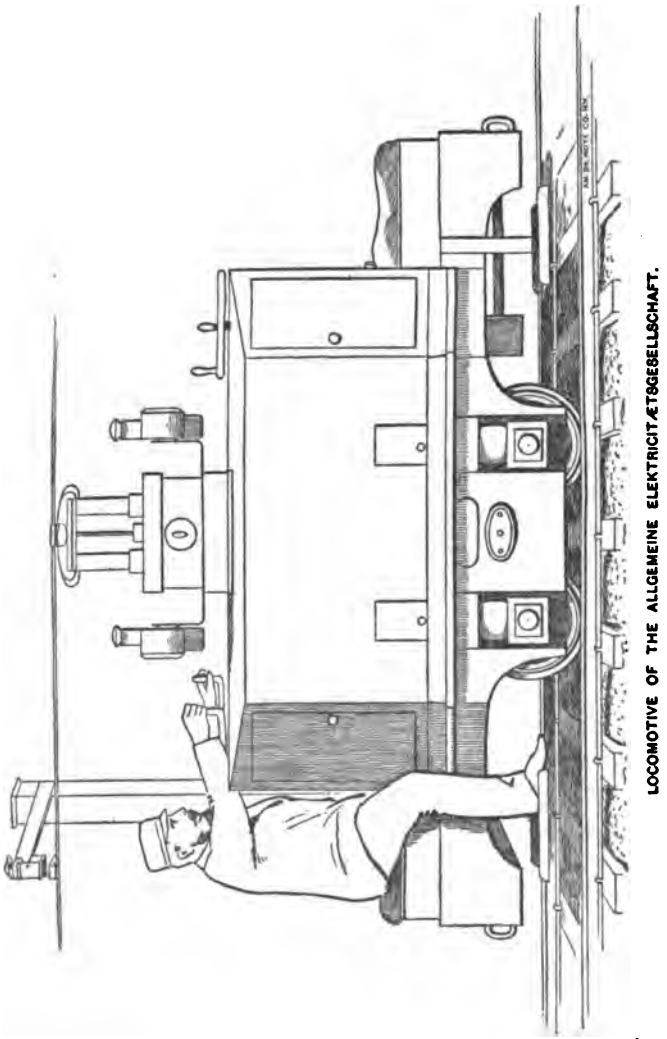


FIG. 4.—LOCOMOTIVE OF THE ALLGEMEINE ELEKTRICITÄTSGESELLSCHAFT.

The plant of the *Allgemeine Elektricitätsgesellschaft* (Fig. 4) shows some differences. There the primary dynamo is fastened directly to the shaft of a 50-horse-power vertical steam-engine, making 250 revolutions per minute. The dynamo is shunt-

wound, with an armature on the principle of the Gramme ring. The collectors are not of copper, but are small blocks of gas-carbon, pressed against the commutator by springs. These brushes are employed because they do not destroy the commutator as rapidly as do those of copper. In this case they are said to have proved exceedingly satisfactory.

From the brushes, the copper cables run to the switch-board (where there are a volt-meter, an ammeter, a cut-out, and lead safety-strips), and thence down the shaft, in which, at intervals of 4 meters, they are supported by porcelain insulators fastened to the shaft-walls.

In the gangway, one of these cables is connected to the rail and the other is in contact with the overhead-conductor, which consists of two strips of band-iron riveted together, or of small profile iron, and is supported at intervals of 6 meters by porcelain insulators.

These insulators are either arranged in the gangway, as shown in the print, or they are fastened directly to the roof.

In this way the conductor is placed directly above the middle of the track, and at a distance above the rail varying between 1630 and 1800 mm.

It is very easy to arrange for a switch with this conductor by simply riveting a branch to the main conductor wherever there is a switch in the track below, no complicated switch in the conductor itself being necessary.

The print shows also the method of taking the current from the upper conductor.

Two springs fastened to the top of the locomotive press a solid block of metal against the band-iron from below, and so allow the electricity to enter the locomotive. There the current is divided, part going to excite the magnets and part flowing through the armature. The motor is shunt-wound, like the dynamo, and the brushes of the motor, like those of the dynamo, are of gas-carbon.

The electric return-current to the dynamo passes through the rail.

To make the comparison more easy, I have arranged the most important data concerning these plants in the form of a table (p. 367).

As to the cost of running, I take the following table from a pamphlet published by Siemens & Halske in 1887 :

Comparison of the Tramming, and the Cost of Tramming with Electric Motors, at New Stassfurt, Zaukeroda, and Hohenzollern.

	New Stassfurt.	Zaukeroda.	Hohenzollern.
Average tramming distance, meters,	800	720	756
Maximum number of cars per shift,	400	400	900
Weight of coal per car in kilogrammes,	800	475	550
Ton-kilometers per car,	0.640	0.342	0.406

Cost per Car, in Pfennigs (1 Pfennig is About ¼ Cent).

	New Stassfurt.	Zaukeroda.	Hohenzollern.
Machinist above ground,	0.9045	0.3120	0.6110
Trainmen under ground,	2.2000	0.8000	.....
Coal for steam,	1.7900	0.3400	0.2230
Various materials,	0.5685	0.6735	0.3330
Repairs,	0.3276		
Interest and depreciation at 15 per cent.,	2.4856	1.0150	1.7780
Total,	8.2762	3.1405	2.9450

Cost per Ton and Kilometer, in Pfennigs.

	New Stassfurt.	Zaukeroda.	Hohenzollern.
Machinist above ground,	1.4133	0.9122	1.5000
Trainmen under ground,	3.4375	2.3392	
Coal for steam,	2.7970	1.0000	0.4570
Various materials,	0.8889	1.9693	0.4010
Repairs,	0.5119		
Interest and depreciation 15 per cent.,	3.8828	2.9678	4.3800
Total,	12.9314	9.1885	6.7380

Cost of Tramming when Done by Man-Power, in Pfennigs.

	New Stassfurt.	Zaukeroda.	Hohenzollern.
Car per shift,	18	7.2	7.5
Car per 100 meters,	2.7	1.0	1.0
Per ton and kilometer,	34.2	21.0	18.0
Ratio of cost of electric-power to cost of man-power,	0.38	0.44	0.37

Cost of Tramming with Horses.

	New Stassfurt.	Zaukeroda.	Hohenzollern.
Per ton and kilometer in pfennigs,	16	12.2	10.0
Ratio of cost of tramming with electricity to that when using horses,	0.75	0.75	0.67

At the mine at Zaukeroda, the new locomotives are very much more efficient than the old, and Mr. Georgi, Bergverwalter, informs me that they expect, on that account, to reduce expenses by 10 per cent.

From New Stassfurt I hear of a reduction even greater. Mr. Hoben, the superintendent there, has kindly sent me a statement of

the expenses of running the electric plant during the month of July, 1890, in marks (one mark being about 25 cents):

	Marks.
Locomotive-drivers,	341.68
Switchmen,	312.64
Machinist above ground,	168.94
Packing and oiling,	173.64
Depreciation, 10 per cent. per annum on 40,000 marks,	333.33
Interest, 4 per cent. per annum,	133.33
Coal,	328 05
Total,	1791.61
Number of ton-kilometers run,	22,125
	Pfennigs.
Cost per ton-kilometer,	8.0
Cost when using man power (as above),	34.2

The fuel consumed per ton-kilometer was 0.05 hectoliters of lignite.

The *Allgemeine Electricitätsgesellschaft* also published, after their experiment at Richterschacht I., an estimate of the cost of running such a plant, based on a length of road of 1000 meters, and using 1 locomotive, a second being kept in reserve. Steam was supposed to be furnished from existing boilers, so that the capital invested in the new plant would amount to only 50,000 marks. It was assumed that 1200 carloads, each of 650 kilogrammes of coal, were to be moved daily. The work would then have been 780 ton-kilometers per diem, or, for the year, 234,000 ton-kilos. For this the yearly outlay was reckoned (steam being supposed to cost 2 pfg. per horse-power per hour) as follows, in marks:

	Marks.
For generation of steam,	1380
Materials for greasing machinery,	600
1 machinist above ground,	1200
3 locomotive-drivers,	3600
Depreciation at 8 per cent. and repairs,	4000
Interest at 5 per cent.,	2500
Total,	13,280

which would make the expense per ton-kilometer only about 5.7 pfennigs.

The best answer to the question whether these plants are satisfactory, is, perhaps, the statement that at each of these collieries supplied by Siemens and Halske an increased number of locomotives has very recently been put into operation. At Zaukeroda

	Zaukerode.	Hohenzollern.	New Stassfurt.	Laurahütte.
Steam Engine.				
Horse power,.....usual	12 (?)	23 } 1 train 43 } 2 trains	25	12 usual. 60 max.
Revolutions per minute,.....usual	225 to 250	34 } going. 34 } going.	50	250
Connection with dynamo,.....	Belt.	Belt.	Belt.	Direct coupling.
Dynamo.				
Type,.....	Main current.	Compound.	Compound.	Shunt.
Armature,.....	Hefner-Alteneck.	Hefner-Alteneck,	Hefner-Alteneck.	Gramme Ring.
Brushes, ...	Lamellæ of copper.	Lamellæ of copper.	Lamellæ of copper.	Gas-carbon.
Ampéres,.....	75 } With	34 } for 140 } amp.	42	34
Volts,.....	150 – 160 } full	320 } merly. 300 }	350	220
Revolutions per minute,.....	700 – 800 } train.	800 to 1000	650	250
Length of each cable,.....meters	260	230	410	190
Motor.				
Type, ...	Main current.	Main current.	Main current.	Shunt dynamo.
Armature, ...	Hefner-Alteneck.	Hefner-Alteneck.	Hefner-Alteneck.	Gramme Ring.
Brushes,.....	Copper.	Copper.	Copper.	Gas-carbon.
Horse-power,.....	6.4	11	8	14 (?)
Weight of locomotive,.....kilogrammes	Old, 1600; new, 2200	2086	2174	3500
Tractive force = $\frac{1}{10}$ of weight,.....	" 160; " 220	200 kg.	200 kg.	350 kg.
Usual load,.....kilogrammes	11,000	20,000 (new).	20,000	23,000
Velocity,.....meters per second	2¼ to 3	3	3	3
Train resistance,.....per 1000	.....	12 to 15	10 (?)	15
Length of road,..... meters	750	1000	900	800
"	.....	800	650	.....
Dimensions of locomotive,...millimeters	2430 × 800 × 1500	2800 × 940 × 1392	2670 × 930 × 1500	2800 × 840 × 1680
" millim. (new)	2800 × 740 × 1500	.....	.....	.....
Track-gauge,.....meters	0.566	628	628	.....
Distance between axles,.....meters	.....	650	480	.....

there are now two; at the Hohenzollern mine four; and here an attempt is to be made to let the trains run in a continuous circuit, so that more than one locomotive may be running in the same drift at the same time. In New Stassfurt there are now three gangways where locomotives are in use, and the company owns four locomotives.

It is true that the plant of the *Allgemeine Electricitätsgesellschaft* has been removed from Richterschacht I.; but I hear from disinterested authority that the removal was due rather to the lack of interest shown by the officers of the mine in trying to overcome such difficulties as always present themselves with new arrangements. Upper Silesia has, however, very recently put in still another electric road, built by Fahmeyer & Co., of Aix-la-Chapelle, at the Aschenborn shaft of the Gottessegen mine. This is said to give very great satisfaction. The locomotive draws a train of 16 cars, each weighing 1000 kilogrammes, at the rate of 3 meters a second. Further details I could not obtain in the time at my disposal.

It may be interesting to note that although all but one of these mines are coal-mines, no precautions had to be taken to prevent gas-explosions, because in none of them is there any development of gas.

Further details concerning the plant and operations at Zaukeroda will soon be accessible in a publication by the officials of the mine in the *Jahrbuch für das Berg- und Hüttenwesen im Königreiche-Sachsen*, which will appear toward the end of this year.

In conclusion, I wish to express my sincere gratitude for the kindly reception with which I have been met, and the friendly manner in which all my questions have been answered and all explanations have been given by the directors and officers of each of the mines above named.

Discussion.

In connection with the foregoing paper, Mr. H. C. Spaulding, Boston, Mass., presented photographs of an electric mine-locomotive and an electric mine-pump, recently constructed by the Thomson-Van Depoele Electric Mining Company. The locomotive, believed to be the largest in the United States, has 60 horse-power capacity; gauge, 3 feet; armature speed, 1000 revolutions; total weight, 21,600 pounds; total height above rails, 3 feet 3½ inches; total width, 6 feet 3½ inches; length of frame, 11 feet 5½ inches; length over all, 12 feet 6½ inches; diameter of wheels, 32 inches. The pump was designed to deliver 100 gallons a minute for a lift of 1500 feet. The water-end was furnished by the Knowles Company. Under actual conditions of 680 pounds pressure, and delivering the maximum amount of water, the pump shows an efficiency of 74 per cent. of the electric energy furnished. The voltage used is 220, and, for protection against moisture, the armature is covered with mica, and the fields are made as nearly water-proof as possible.

NOTES ON THE IRON-ORES OF DANVILLE, PENNSYLVANIA, WITH A DESCRIPTION OF THE LONG-WALL METHOD OF MINING USED IN WORKING THEM.

BY H. H. STOEK, SOUTH BETHLEHEM, PA.

(Glen Summit Meeting, October, 1891.)

DANVILLE, the county seat of Montour county, Pa., was one of the earliest and best-known centers of the iron industry in the State. It is situated on the north bank of the north branch of the Susquehanna river, twelve miles east of the confluence of the north and west branches of that river at Northumberland. The ore-belt, extending for a distance of twelve miles along the north bank, has been developed in a number of places, but nowhere so extensively as in the immediate vicinity of Danville. Hence, a description of the mines in that locality will suffice, as typical of the workings of the entire section.

GEOLOGY.

The Montour ridge consists of an anticlinal fold extending from the source of Wapwollopen creek on the east to the west branch of the Susquehanna, which it reaches at the mouth of Chilisquaque creek. According to the reports of the First Geological Survey of Pennsylvania, the length of the ridge is nearly twenty-seven miles, with an average breadth of three-fourths of a mile, and a maximum elevation of six hundred feet in the vicinity of Danville, where it also attains its greatest width. Just north of Danville the Mahoning creek cuts through it, exposing the iron-ore on the hill-side, where advantage has been taken of this to work the beds on the southern slope of the ridge by means of drifts.

The sections given in Fig. 1 are taken from the reports of the First Geological Survey of Pennsylvania, made in 1858, by H. D Rogers, whose description of the Montour ridge was so complete and accurate that the entire report was reprinted as an appendix to Vol. G 7 of the Second Geological Survey. The approximate cross-section shows that near Danville the top of the ridge is of Clinton

slate, flanked by a vein of block- or sandstone-ore from 18 to 24 inches thick. Above this is an iron sandstone covered by a green slate containing thin plates of argillaceous sandstone. Next we find the lower calcareous shales, in which is the fossil ore-bed, from three to four feet thick. Going eastward along the ridge, the strata dip under the coal-measures, and in the vicinity of Bloomsburg the block-ore does not outcrop, but arches the ridge at some distance beneath the surface. The cap of the ridge has been evidently eroded and transported, as masses of these ores are found scattered irregularly over the country to the southward. As may be easily seen from the section, the dip of the measures upon the southern slope is less than upon the northern. On the south side the dip is from 20 to 35 degrees, while on the north side it varies from 30 to 70 degrees, thus giving on that side a much shorter working breast above water level. The general strike of the beds is nearly east and west.

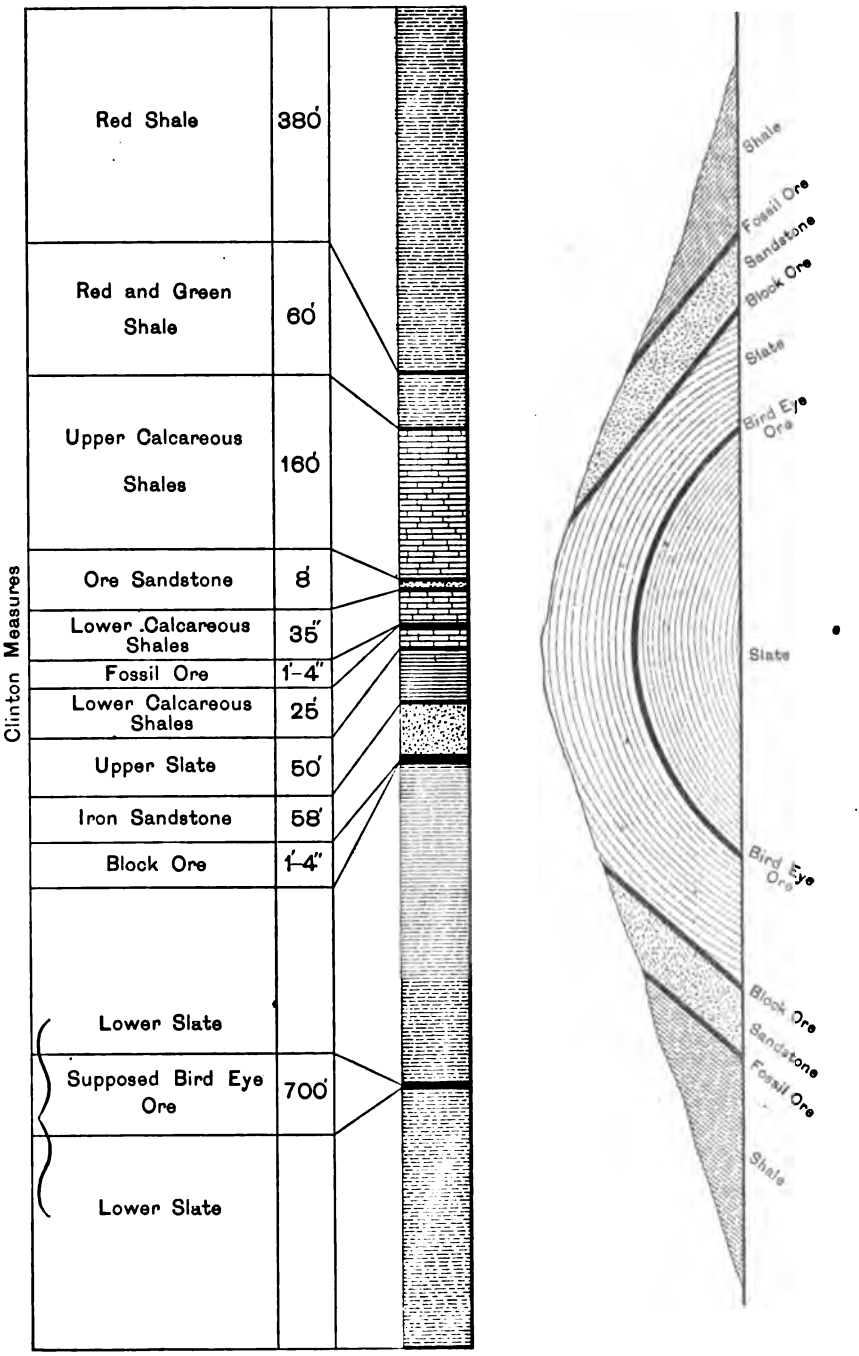
History.

The first development on the south side of the ridge in the vicinity of Danville was made in 1839–40 by David L. Leavitt, of New York city, who was succeeded, in 1843, by Chambers, Biddle & Co., under whose management the first blast-furnace in Danville was built. This firm controlled the mines until 1851, when the Montour Iron Company was incorporated by the Grove Brothers, who retained the ownership until 1857. In that year the company failed, and in 1858 Messrs. Waterman and Beaver secured control of the plant, and it was during their ownership, from 1858 to 1878, that the greatest development was made in the mines. In 1878, Mr. Beaver retired from the firm, and for two years Mr. Waterman retained sole ownership. At the end of this time the furnaces and mines of the Montour Iron Company became the property of the present owners, the Philadelphia and Reading Railroad Company, though the name of the plant has not changed.

The Ores.

The ores are of two distinct varieties, the limestone-, or Danville fossil-ore, and the sandstone-, or block-ore. The fossil-ore is simply a fossil-limestone impregnated with iron. The bed is usually divided into three splits by partings of sand-slate. Beneath a slate top we find the upper split from 5 to 6 inches thick, under which is a parting of 6 to 8 inches of sand-slate. The middle split, known

Fig. I.



SECTIONS OF MONTOUR RIDGE AT DANVILLE PA.

as the "buncombe," is the main bed, and varies from 15 to 20 inches in thickness. Below a parting of ten to twelve inches of sand-slate there is sometimes found a thin split of ore, at times reaching a thickness of 5 inches, with a bottom rock of slate.

The "buncombe" is the richest of the three splits: the others contain more slate and their analyses show larger percentages of lime and magnesia.

The fossil-ore is divided into two kinds, the "hard" or limestone-ore, and the "soft ore." The latter occurs in the form of a pasty mud, which has been derived from the hard ore by the leaching out of most of the lime. There seems to be no regularity in the occurrence of these two ores, hard ore changing into soft, sometimes gradually, but very often suddenly. In many places, hard and soft ores may be found within a few inches of each other. While the top rock of the hard ore is usually a solid slate, that of the soft ore is cracked and broken. The fossil-ore extends along the southern slope of the ridge for two miles to the east of Mahoning creek, where it pinches out, and for 5 or 6 miles the bed is not workable; but it then reappears and extends to Bloomsburg, 12 miles from Danville, where it is again worked. On the north slope of the ridge the fossil-beds are regular on the east side of Mahoning creek, but toward the west the parting gradually increases in thickness until it replaces the ore.

The Narrows of the Susquehanna, two miles west of Danville, is stated by the State Geological Survey to be the western limit of economical working for these beds. Their development, however, had not been pushed so far when work was stopped.

The block-ore is a sandstone highly impregnated with iron, and breaking when blasted into blocks, whence its name. Near the outcrop it is very rich; and through the use of this ore, which makes an excellent blast-furnace mixture with the fossil-ore, Danville iron acquired its reputation for superior quality. The beds have an average thickness of 18 to 24 inches, and wherever worked they have been very regular on both the north and south slopes of the ridge. Both the foot-wall and the hanging-wall are of sandstone.

The Second Geological Survey locates, between the block- and fossil-ores, a bed of sandstone ore which is said to extend to the west of Mahoning creek, but not on the east side. As there is no record of attempts to develop this ore, it is probably of little economic importance compared with the other two.

In the counties west of Montour county, the valuable "bird-eye"

ore is found below the block-ore, and 150 feet above the contact of the Clinton and Medina measures. This same ore outcrops at Chaulaski, three miles below Danville, on the north bank of the Susquehanna, but only a small amount of ore has been taken out, the outcrop being but 5 inches thick. Borings have not been made in the Danville measures to search for this ore, and there is a diversity of speculative opinion as to its probable occurrence. According to some of those who are working the ore west of Danville, the bed pinches out toward the east; and if this be true, an outcrop of only 5 inches at Chaulaski might be taken as a possible indication that at Danville the bed, if present, is not of workable thickness, though of course, in the distance of 3 miles, the bed may have again become workable.

Until test-borings are made, speculation will probably continue as vigorous and as uncertain as at the present time.

The following analyses of the ores have been made at various times during the last ten years in the laboratory of the Montour Iron Company.

Soft Fossil-Ore.

	1882.	1889.
H_2O,	3.50	7.05
SiO_2,	19.28	24.11
Fe_2O_3,	67.30	60.49
Al_2O_3,	7.94	2.45
CaO,	0.86	1.12
MgO,	0.80	0.43
S,	0.02	0.02
P,	0 47	0.70
MnO,	.....	3.20
CO_2,	.....	.....
	100.17	99.57

Hard Fossil-Ore.

	Pursel Slope. 1888.	Welsh Hill. 1889.
H_2O,	.....	0.18
SiO_2,	5.56	6.04
Fe_2O_3,	38.81	28.65
Al_2O_3,	4.34	6.95
CaO,	25.17	29.44
MgO,	2.85	2.17
S,	0.10	0.60
P,	0.23	0.15
MnO,	.....	.....
Co_2,	23.00	24.66
	100.06	98.84

The soft fossil-ore as it comes from the mines contains 10 to 20 per cent. of water, but only 7 per cent. after drying on the ore-heap for one year. Eight analyses of the fossil-ore from Welsh Hill drift gave a mean of 21.89 per cent. of iron. A complete analysis of the block-ore could not be obtained. Its composition is very similar to that of the soft fossil-ore, excepting a smaller percentage of water and a higher percentage of silica.

The following analyses were made in 1882 by Messrs. Booth, Garret and Blair:

	Hard Fossil.	Soft Fossil.	Block.
SiO_2,	19.28	5.56	27.28
Fe_2O_3,	64.65	39.10	52.23
Al_2O_3,	7.93	4.30	8.29
CaO,	0.86	26.17	2.43
MgO,	0.79	2.60	1.54
S,	0.22	.....	0.17
P,	0.46	.....	1.56
Mn,	trace.	.....	trace.
Co_2,	.....	22.81	.....

The iron made from the ores in the western part of the Danville workings was cold-short, while as the workings advanced towards the east it became neutral, and then red-short. For the sake of comparison the following analysis of the "bird-eye" ore from Snyder county is added:

Fe_2O_3,	67.81
SiO_2,	11.38
CaO,	12.64
MgO,	4.02
Al_2O_3,	2.09
MnO,	1.30
S,	0.16
P,	0.41
	99.81

Development and Present Condition of the Mines.

The first mining was done in the block-ore north of a line drawn west from the slope A (Fig. 2). Drifts were driven westward to the property line and the ore worked out to the outcrop. In 1853 the slope A was sunk, and with later extensions a depth of nearly 500 feet was reached, thus tapping the fossil iron-ores at perhaps the greatest depth below water-level yet reached on these ores in this country. Six levels were driven to the west and the ore was worked

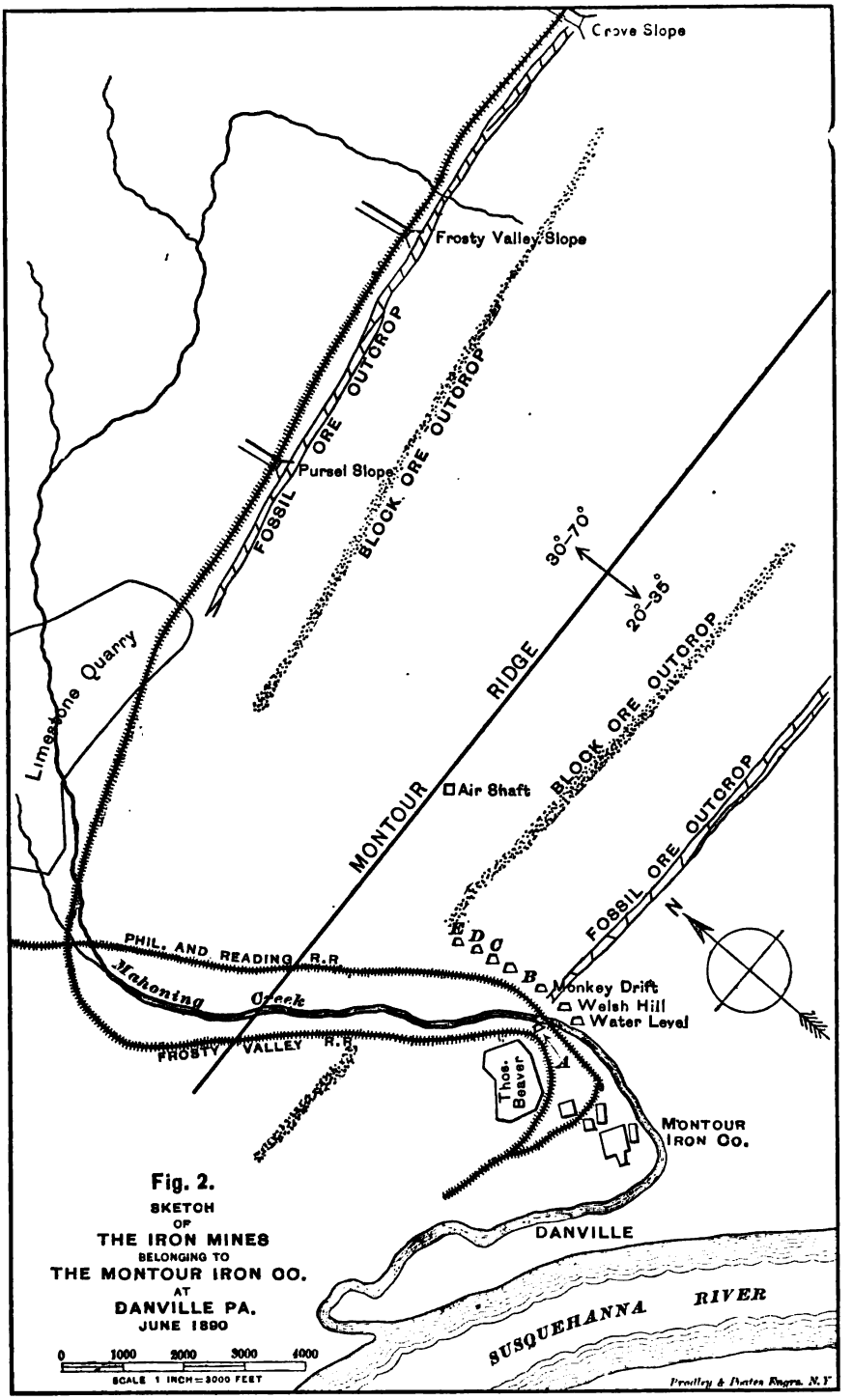


Fig. 2.
SKETCH
OF
THE IRON MINES
BELONGING TO
THE MONTOUR IRON CO.
AT
DANVILLE PA.
JUNE 1890

out to the property-line, while two levels were started to the east, but soon abandoned, as the ore became too lean to be profitably worked. Owing to the constantly increasing expense for pumping, the difficulty of ventilation, and to the large amount of ore above water-level owned by the company, the slope was abandoned many years ago, and is now full of water, the only evidence of former activity being the wooden engine-foundations which are still in place. On the east side of Mahoning creek two drifts were driven in the fossil-ore, the "Water Level" and the "Welsh Hill" drifts (Fig. 2). About 2500 yards of gangway were driven in this bed, as shown in Fig. 3, and when the work was stopped in 1889 most of the ore between the gangways had been taken out. In the block-ore, higher up on the hillsides, five drifts have been driven, the "Monkey" drift, about 500 yards long, and B, C, D and E, approximately as shown in Fig. 4. No new drifts have been started since 1880, but most of the ore has been taken out between the gangways. On the northern slope of the ridge the outcrops were worked prior to 1850, to supply ore for a small charcoal blast-furnace, which was long ago destroyed.

Soon after Messrs. Waterman and Beaver secured control of the Montour Iron Company, about 1860, the Pursel and Frosty Valley slopes were sunk in the fossil ore-bed (see Fig. 2.). The Frosty Valley slope (Fig. 5) was sunk about 400 feet upon a dip of 35 to 45 degrees. Gangways were driven east and west 75 feet apart, thus forming five lifts worked from this slope.

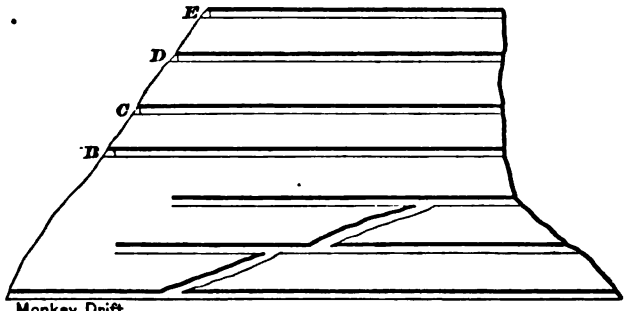
Numbering the lifts from the top downward, lifts 1 and 2 were worked out on the east to the property-line, and on the west until they connected with the Pursel slope-workings.

Lifts 3, 4 and 5 had been partly worked out when this slope was abandoned in 1889, as shown approximately in Fig. 5. Most of the ore taken from this slope was of the soft fossil variety.

The Pursel slope is located on the same outcrop, three-fourths of a mile west of the Frosty Valley slope. It was sunk about 600 feet upon a dip of 25 to 35 degrees, giving six lifts of 90 feet each and a sump at the bottom. The water of this slope was pumped to the gangway of the highest lift, which was pushed to the surface and used as a drainage-adit.

Lifts 1 and 2 had been worked out, and the remaining lifts partly developed, when the slope was abandoned in 1889 and allowed to fill with water. The Grove slope upon the property of the Messrs. Grove, three-quarters of a mile east of the Frosty Valley slope, was

Fig. 4.



BLOCK ORE

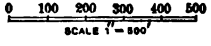
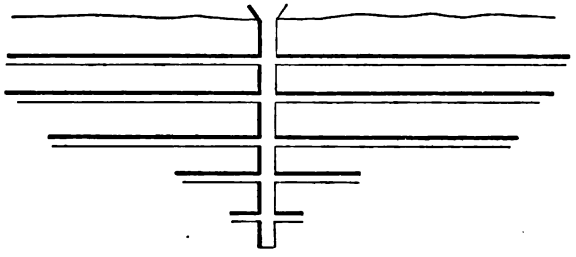
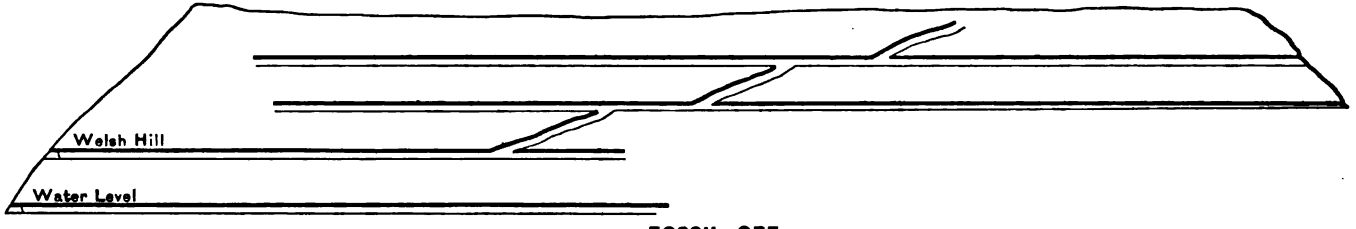


Fig. 5.



FROSTY VALLEY WORKINGS.

Fig. 3.



FOSSIL ORE

abandoned in 1882, after which time the amount of water to be raised through the Frosty Valley and Pursel slopes was greatly increased.

In September, 1889, all work was stopped in the mines of the company, on account of the constantly increasing cost of mining upon the north side of the ridge, due to the large amount of water to be raised, and on account also of the small percentage of iron contained in the ore. Since the mines were stopped the furnaces of the company have been run largely upon Lake ores. Excepting in the Monkey drift upon the south side, all machinery and rails were removed and the mines permanently abandoned.

In that drift the rails are still in place, and work can there be resumed at any time, should it be necessary to do so, to supply the two furnaces of the company with ore.

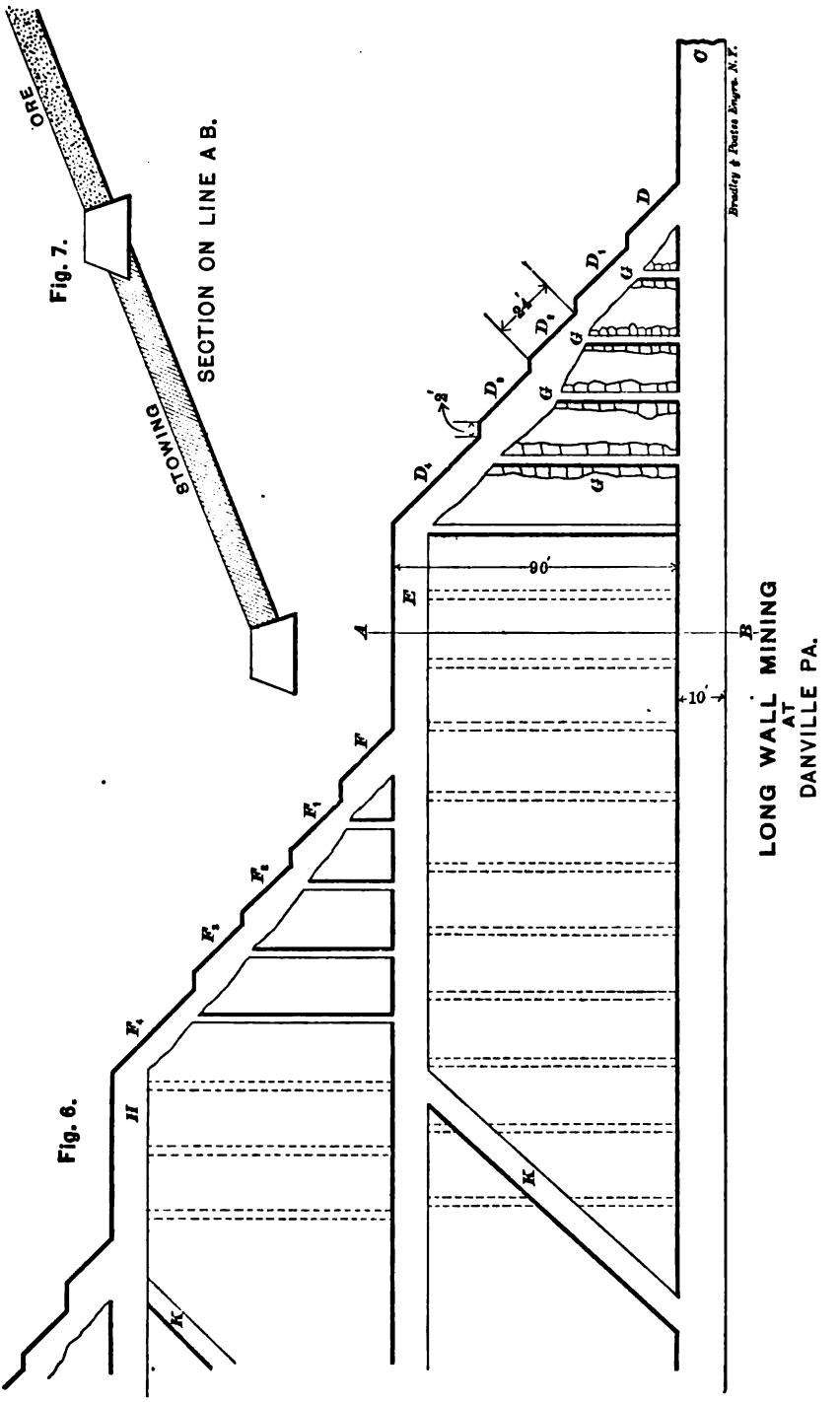
When visited, in June, 1890, the mines had been abandoned for a year; the slopes were full of water; and the Monkey drift was the only one into which it was considered safe to venture. As the workings had not been mapped, the accompanying sketches merely approximate the extent and general arrangement of the mines at the time of their abandonment. They were drawn as accurately as possible from descriptions furnished by former mine-foremen, and information obtained from a number of persons who were officially connected with the mines at various times.

Method of Mining.

The long-wall method of mining was introduced into these mines by Mr. Daniel Edwards, formerly Superintendent of the Montour Iron Company, but now President of the Kingston Coal Company. At the time of its introduction it was a method but little used in this country, and seldom in beds as thick as those of Danville, where the breasts were frequently 4 to 5 feet high. Figs. 6 and 7 show the general plan of the method as used at Danville.

Levels are driven 90 feet apart, and the face of each gangway should be kept in advance of all higher gangways, so that of the gangways, C, E and H, for instance, the face of C should be the farthest from the slope or the mouth of the drift; but it will be seen by reference to Fig. 3 that this rule has not been carried out. In fact, this is seldom done in this region, as it necessitates the outlay of a large amount of capital before any return.

The gangways are driven 7 to 10 feet wide and 5½ to 7 feet high, *i.e.*, just high enough for a man or mule to go erect. The lowest


ORE
STOWING
SECTION ON LINE A B.
Fig. 7.
LONG WALL MINING
AT
DANVILLE PA.
Fig. 6.
90'
10'
A
B
E
F
F1
F2
F3
F4
H
K
K1
D
D1
D2
D3
D4
G
O
Bradley & Poates Engrs. N.Y.

gangway, C, Fig. 6, is known as a "fast-end" gangway, as it is driven entirely in the solid, while E, H, etc., are "loose-end" gangways, with but one side in the solid and the other side formed by the stowing. The face of the gangway, C, should be kept far enough ahead so that blasting there will not interfere with the workmen in the breast, D, and the same consideration should determine the distance of the breast, F, from the face of E.

To facilitate the loading of the ore into cars from chutes, the gangways are so driven that the bed will lie at the top of the upper rib, Fig. 8, and to secure the proper gangway-height the bottom rock must be taken up. In this point the method differs from that followed at Mansfeld, Germany, where the top rock is blown down. In doing this along the lower gangway, C, a drainage-ditch is left upon the lower side. When first driven, the gangway, C, is timbered upon the upper side, as shown in Fig. 8; but, as settling takes place, the props are usually broken, and it is generally necessary to renew them and to blow down the top of the gangway, which frequently settles sufficiently to obstruct the haulage-way (see Fig. 9). Often, however, the stowing becomes so tightly packed in settling that retimbering is unnecessary. In the soft ore, by reason of the creeping of the bottom, the gangway-props must sometimes be renewed several times.

Breasts, D, D, etc., F, F, etc., are turned off at an angle of 35° to 45° with the direction of the gangway, the angle being determined by the dip of the bed. The breasts are 2 feet wide and 24 to 30 feet long, and usually there are five breasts in a tier between two gangways. The height of the breast varies, with the nature of the ore, from 2 to 5 feet. In the hard limestone-ore the three splits are taken out if the ore in the top and bottom splits is sufficiently rich; but if these splits are thin, or if the ore is lean, the "buncombe" merely is mined, with enough of the top to allow a miner to square his shoulders in the breast. In the soft fossil-ore only the "buncombe" is taken out, with enough rock to allow the miner to work his breast. The hard limestone-ore and the block-ore have to be blasted, but the soft ore is scraped out in the form of mud.

Each breast is worked by a miner and one laborer; or two miners will combine and work two breasts; and, sometimes, one miner and two or three laborers will work two breasts. The miner working the top breast of a tier, such as D_4, Fig. 6, also drives the gangway, E, takes up the bottom rock to give sufficient height for haulage, piles the stowing carefully on the lower side of the gangway, and

prepares the road-bed for the track-layers, for which additional work he is paid extra.

A ditch is not left along the loose-end gangway, as the water should drain through the stowing to the fast-end gangway, C. The gob is thrown loosely between the breast and gangway below, excepting along the chutes, G, G, where it is carefully piled in the form of a dry wall to support the roof and to protect the chutes. A chute 2 feet 4 inches wide is left for each breast, down which the ore is thrown to the gangway below. It is sometimes necessary to line these chutes with boards, though generally a carefully-built dry wall of gob suffices. The ore from each breast is carried to the chute by hand, and is allowed to accumulate at the bottom of the chute, being drawn out into cars as desired.

There is usually a platform at the bottom of the chute to facilitate the loading; in the soft ore it is placed directly above the car (Fig. 10), but it is nearer the bottom in the hard ore, and sometimes the ore is simply allowed to pile up along the gangway. When necessary to prevent the air-current drawing up a chute, a canvas curtain is hung loosely over the mouth of it (Fig. 11); but ordinarily this is unnecessary, as only the last five inside chutes are kept open, the others being boarded up and filled with gob when the gangway, E, has advanced so that the ore from the next higher tier of breasts can be removed through that gangway. The gangways, E, H, etc., are connected with the fast-end gangway, C, by "pitching gangways" K, K, driven through the gob; and back of these connecting-roads the gangways, E and H, are abandoned, so that the fast-end gangway, C, is the only one kept open through its entire length. The gangways in the soft ore are timbered and lagged on sides and top, but in the hard limestone-ore and in the block-ore it is generally necessary to timber the sides only, as the roof is of good slate or sandstone.

The props are placed 2 to 6 feet apart, the distance depending upon the nature of the roof. In the hard fossil-ore and in the block-ore the breasts are not timbered, excepting when necessary to protect the chutes, as the gob fills up the space and supports the top. In the soft fossil-ore small props, 3 to 5 inches in diameter, are used to keep up the top, as the gob does not fill more than one-third of the vacant space. Heavy timbers are usually placed along all chutes. The timber is furnished by the company, but the miners set it, both in the breasts and along the gangway; and as it

is cut on the property of the company, the expense of timbering is very small.

All general work, such as track-laying, the clearing away of "falls," etc., is done by miners, detailed for each separate piece of work, instead of by "company hands," as is ordinarily done, and for such work the miners are paid per diem. The machinery used about the mines is generally of the simplest kinds. All drilling is done by hand, and the ventilation is secured by natural draft through chimneys.

In the drifts, the cars are either pushed by hand or hauled by mules to the mouth, where the ore is dumped into wagons and hauled down the mountain by teams to the furnace stock-pile, or to the drying-pile.

In the slopes, the mine-cars are hoisted to the top by second-motion engines, and then run to a platform above the tracks of the Frosty Valley railroad and dumped into the cars on the track below by means of a chain, which is hooked to the back end of the car, and raised by a vertical hand-windlass. Fig. 12 shows the device used for turning the mine-cars from the gangway on to the slope. One of these shifting turn-tables was used at each lift along the slope. It consists of a flat car, A, on which is a turn-table, the car when not in use being run into the room, B, cut out of the top-rock of the slope.

The car is run into the position, A′, over a bridge, C, C′, the end, C′, of which can be raised and lowered by means of hinges placed at C, so that when not in use the bridge is raised, and does not interfere with traffic on the slope. The tracks on the bridge, when in the position, C, C′, are below the level of the gangway tracks, so that with the car at A′, the tracks on the car connect with the gangway tracks. The tracks running lengthwise along the car A are turned up at the end and flattened, so that, when at A′, they connect with the tracks along the slope. To use the turn-table, the bridge is lowered, the car A run into the position A′, and the mine-car run from the gangway on to the table, which is turned through 90 degrees, thus bringing the car into the direction of the slope, when the hoisting-rope is attached and the car is hoisted to the top. The turn-table is then run back, the bridge is lifted, and the slope is open for hoisting from the lower levels.

The mine-cars are 2 feet deep, 4 feet long, and 3½ feet wide, and hold about one ton of ore, a cubic yard weighing about one ton. The gauge is 30 inches and the wheels are 14 inches in diameter,

Fig. 8.

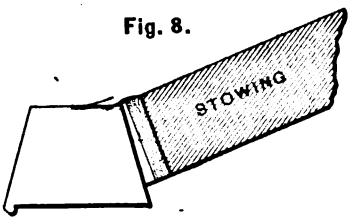


Fig. 9.

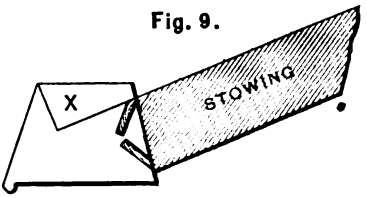


Fig. 10.

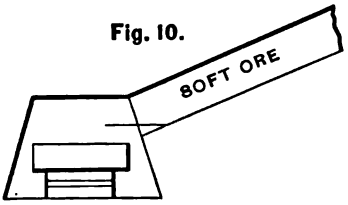
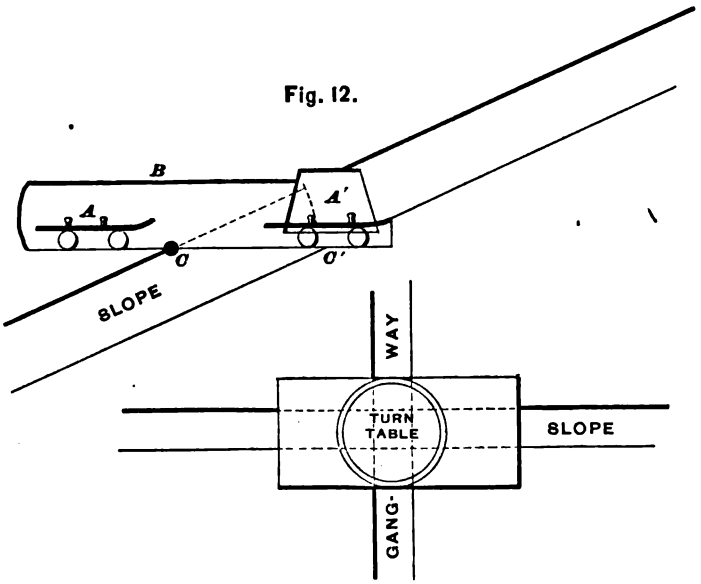


Fig. 11.

Fig. 12.



DETAILS OF UNDERGROUND WORKINGS.

1¾ inches bore, and are loose upon the axle. The slope mine-cars have a gauge of 36 inches, and wheels 15 inches in diameter. The average cost of mine-cars, about 1877, was $50 each.

The Frosty Valley railroad runs from the Grove slope past the Frosty valley and Pursel slopes, past the limestone quarries of the company, and along the west bank of Mahoning creek to the furnaces and rolling-mills of the Montour Iron Company, in Danville, a distance of 5 to 6 miles. The road has a gauge of 40 inches, and was built in 1860 by Messrs. Waterman and Beaver to carry the ore and limestone to the furnaces. The railroad ore-cars hold about 4 tons and are of two kinds, a drop-bottom car for the soft ore and a solid-bottom car for the hard ore.

Cost of Mining.

For breast-work, miners are paid by the ton and for gangway-work they are paid tonnage and yardage.

Tonnage payments depend upon:

1. The nature of the ore.
2. The height of the breast.

Yardage payments depend upon:

1. The nature of the ore.
2. The kind of gangway.

Since the nature of the ore in the fossil-beds and the height of breasts vary so irregularly, it is almost impossible to give exact figures which will have much value; so that chiefly ratios will be given. Upon a basis of one dollar per ton for mining block-ore, the following are about the prices paid during the past 15 years:

Block-ore per ton,	$1.00
Block-ore per yard, fast-end gangway,	4.00
Block-ore per yard, loose-end gangway,	1.70
Hard fossil-ore per ton,	0.95
Hard fossil-ore per yard, fast-end gangway,	6.25
Hard fossil-ore per yard, loose-end gangway,	2.40
Unskilled labor per day,	0.73

Since "soft ore" is a relative term, it might be misleading to state exact values for it, but one-third to one-half the price per ton paid for hard ore may be taken as a fair approximation. Upon the same basis as the preceding, the mixed ores delivered at the stock-house cost $1.37 per ton. During the past 15 years, the price paid for mining block-ore has varied from $2.50 to $1.60 per ton.

One ton per day for each man working in a breast is considered an average output for a shift of 10 hours. The miner pays his laborer, or laborers, per diem, at the above rate. In gangway-work the average rate of advance was 15 feet per month for loose-end gangways and 7 feet per month for fast-end gangways.

Owing to the many conditions affecting the rate of advance along the gangways, it was necessary to employ a system of "allowances" in payment of gangway-yardage so as to equalize as nearly as possible the pay of gangway-miners.

When mining upon leased property, the following royalties were paid per ton of 2240 lbs.

	Per ton.
Hard ore,	$0.25
Soft ore,	0.50
Limestone,	0.12½

For much of the data contained in this paper, I am indebted to Professor Edward H. Williams, Jr., who allowed me to make use of his notes, secured while the mines were still in operation; also to Mr. R. K. Polk, formerly Assistant Superintendent of the Montour Iron Company, and in charge of the mines when they were abandoned, who personally aided me in securing information.

THE UTILIZATION OF PUDDLE- AND RE-HEATING-SLAGS FOR PAINT-STOCK.

BY AXEL SAHLIN, M.E., NEW YORK CITY.

(Glen Summit Meeting, October, 1891.)

SINCE the days of the infancy of metallurgical industries, slags of various kinds have been accumulating. Until a comparatively recent date hardly any employment had been found for these waste products, which have had, consequently, little or no value. An exception to this rule is the basic Bessemer slag, the value of which has become universally recognized. It is to-day one of the standard fertilizers of the world, and a price of from $15 to $20 a ton is considered reasonable for the ground slag, which, especially in Germany, finds a ready market as fast as it is prepared.

The slags from puddling and re-heating furnaces have only found

use hitherto as a low-priced ore. The density and unchangeable nature of these slags have, however, recently opened for them a new, though limited, field, where they can command, in a powdered state, almost twice the price of the famous basic Bessemer slag, namely, as paint-stock. The first experiments with slags for this purpose were made at Pullman, Ill., about five years ago. A plant was built for the grinding and subsequent treatment of slag; patents covering the idea and mode of preparation of slag-paints were secured, and a fair degree of success was attained. Great difficulty was encountered, however, in getting the slag reduced uniformly to sufficient fineness.

Burr-stones and impact-mills of various descriptions were successively tried. While the grinding-parts of the apparatus were new and sharp, a fair article was produced, but, as the parts wore down, the product became coarser and unfit for paint-stock. Later, a very crude machine, on the principle of the Cyclone pulverizer, was introduced with good results. The capacity of the plant was greatly increased, and the product was at all times uniform; but the handling of this class of machinery was so little understood, and the machine was so crude in construction, that the finest dust, which is the most valuable part of the product, was wasted. Nevertheless, a large quantity of serviceable paint-stock was made, and enough attention was called to the new material to cause the paint-trade to acknowledge its value.

For reasons in no way reflecting on the technical success of slag-paint, the Pullman plant was eventually put out of operation, and no more slag was ground until 1889, when a company was formed, the patents were bought, and a plant for the purpose of making slag-paint was erected at Boonton, N. J., where water-power and large deposits of puddle- and heating-cinder were available. Samples of these slags have shown a chemical composition as follows:

	Puddle-slag.	Heating-cinder.
FeO,	52.430	71.290
Fe_2O_3,	19.620	
MnO,	6.410	0.210
CaO,	0.810	.....
Al_2O_3,	0.380	7.780
SiO_2,	16.390	20.060
P_2O_5,	3.840	0.270
S,	.....	trace.

The heating-cinder has so far been preferred on account of its

being in better mechanical shape and cleaner, and also on account of higher SiO_2; but both slags make very good paint-stock.

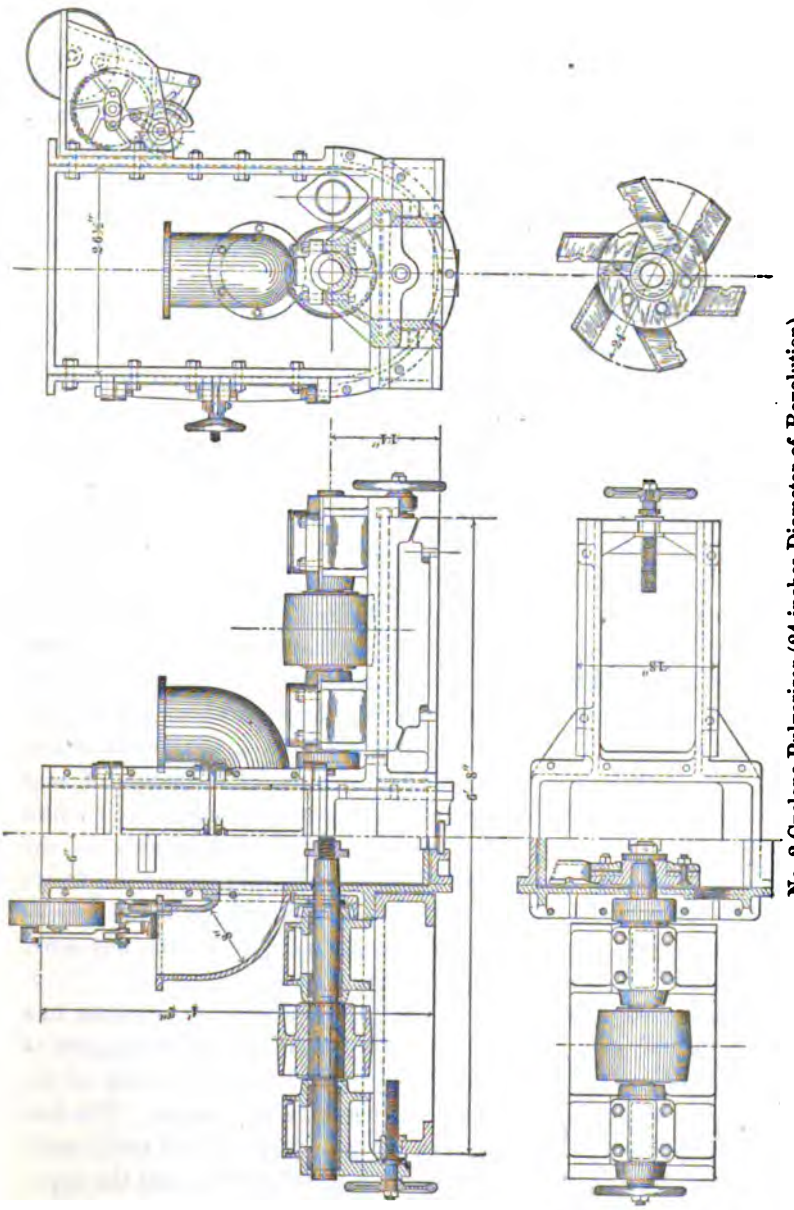


FIG. 1.

No. 2 Cyclone Pulverizer (24 inches Diameter of Revolution).

The slag is crushed in an ordinary Blake crusher to pass a $\frac{3}{4}$-inch

wire screen, and is then conveyed to the grinding-machine by an elevator and an 8-inch screw conveyor. The disintegrating machine is a 24-inch Cyclone pulverizer, shown by Fig. 1. It consists of two 24-inch steel fans, working face to face near the bottom of a steel-lined iron chamber, at a speed of 2400 revolutions per minute. The fan-blades are made of $3\frac{1}{2} \times \frac{5}{8}$ inch Bessemer steel, cut 10 inches long and twisted at the middle to an angle of 35° from the plane of rotation. They are firmly held between two disks, constituting the hub of the fans, and are pitched forward 15° from the radial line, thus having a tendency to throw particles, with which they collide, towards the center of the opposite fan, and partially counteracting the centrifugal force.

The crushed slag is dropped at a uniform rate in the center of the pulverizing-chamber by an adjustable, automatic tumbler-feed, and is caught in the vortex of air-currents created by the revolving-fans. The different pieces are violently dashed or rubbed against each other, and occasionally receive a new impetus by striking against the fans or the sides of the chamber, whence they again rebound and start on their rapid course to destruction and disintegration. A particle of slag suspended between the fans, assuming it to remain motionless, would be passed by a fan-blade with its accompanying air-current 480 times a second, or 28,800 times each minute. The effect can readily be imagined. Quicker than thought the lumps of hard slag are converted into smoke, or dust, so fine that it floats in the air. It will thus be seen that this pulverizer acts mainly by concussion and attrition, but also, to some extent, by impact, which manifests itself by the wear on the fan-blades. The amount of wear on these blades, however, does not in any way influence the fineness and evenness of the product, which is controlled entirely by the velocity of the air passing through the chamber and by the buoyancy of the particles. This circumstance makes the pulverizer above described invaluable for grinding paint-stocks, or, in fact, any other material for which an absolute fineness is demanded.

Nevertheless, it has been important, economically, to reduce to a minimum the wear on the fans, which are the only wearing-parts of the machine periodically requiring renewal. A short review of the materials successively tried for blades may be of interest. The first was wrought-iron. Though very cheap in first cost and easily made up into shape, it wore so rapidly that it was discarded, and the opposite extreme, chilled cast-iron, was substituted. Of all materials ever tried, no other has stood so much wear as this; but its brittle-

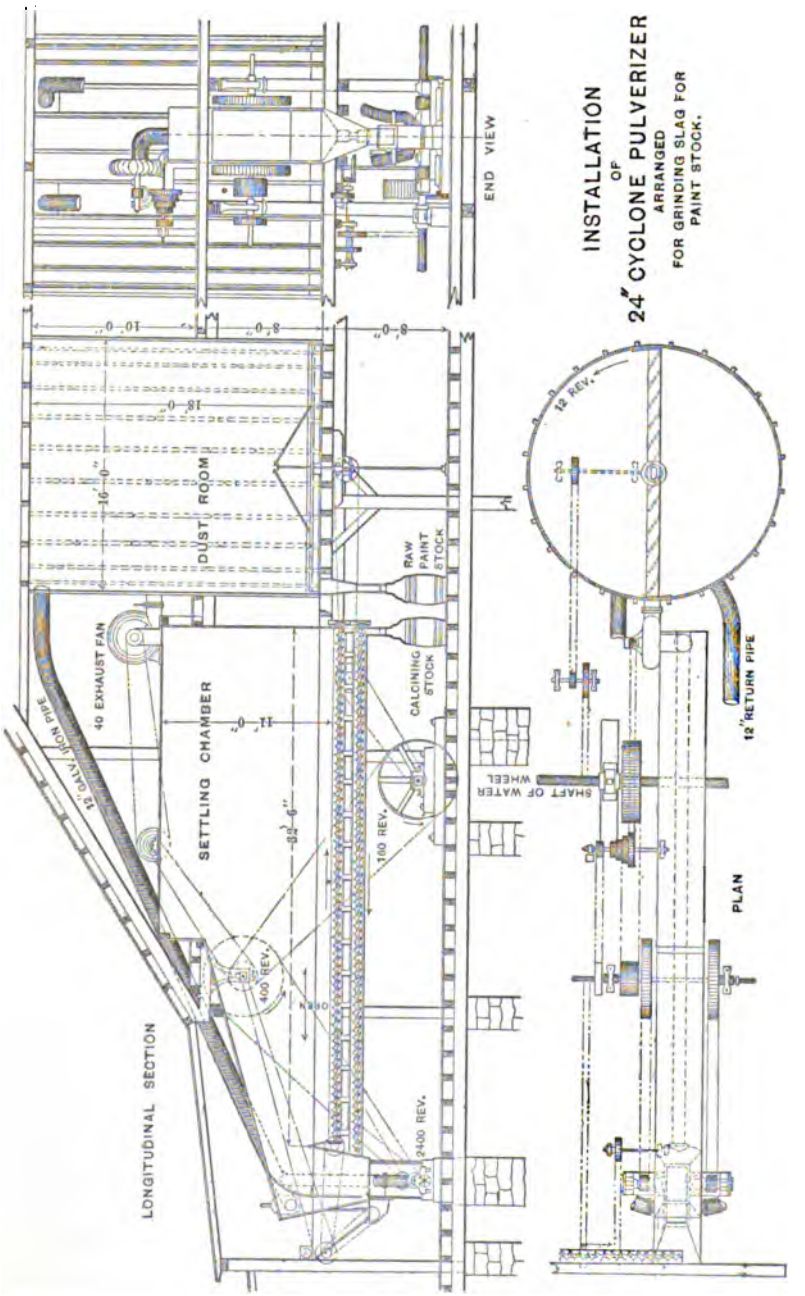


FIG. 2.

ness endangers the machine, since if one blade should break, the whole twelve are very apt to go, and the shafts may be sprung. This

material was, therefore, reluctantly abandoned. High-carbon tool-steels were tried next, being tempered, annealed, and hardened in different ways for different tests, but they have not shown sufficient durability to justify the high initial cost. Chrome-steel has done better, and in fact answers very well for grinding many materials where discoloration has to be guarded against; but it is difficult to handle, and even in cases where it is otherwise satisfactory, its high price is a serious objection. Hadfield steel, containing about 13 per cent. of manganese, has given promise of excellent results, but the samples made at different works in this country have been unreliable and unsound; the only good castings obtained of this wonderful paradox have been imported from England. The Hadfield steel has, in some cases, shown a wearing-capacity almost equalling that of cast-iron; and the fact that car-wheels made of it are reported to last longer than even chilled-iron wheels, gives good ground for the hope, that this material, when better known, will be largely introduced in the place of chilled-iron in crushing- and grinding-machinery in general. At present the price places it out of reach commercially.

Finally, it has been concluded that, for general use, ordinary Bessemer steel, with 0.30 to 0.40 carbon, is about the most economical. On soft materials it stands very well, and on hard materials the cost of renewals is moderate. On raw heating-cinder, one set of blades pulverizes about 28,000 pounds, making the present cost of renewals 17 cents per ton. As the grinding enhances the value of the slag at least $20 a ton, this is not a heavy tax, especially when the excessive fineness of the product and the hardness of the slag are considered. On the calcined slag described below, one set of blades has been in use for nine weeks without renewals.

Fig. 2 shows the arrangement of the plant for feeding the pulverizer and for drawing off, separating, and collecting the dust. A conveyor from the crusher delivers the slag into a hopper placed above the feed-box, whence it is automatically delivered into the pulverizer and ground in the manner above described. Dust-laden air is constantly drawn off from the top of the pulverizer by means of an exhaust-fan, placed at the opposite end of a long and narrow chamber, connecting with the top of the pulverizer. This chamber is 32 feet long, 3 feet wide, and 11 feet high. It is perfectly open and unobstructed from one end to the other. In passing through this space the air gradually drops the coarser dust; the larger particles settle sooner than the smaller ones, and thus is given a means for forming

a separation more perfect than that of any bolting-process. Along the hoppered bottom of the chamber runs a double flight of 6-inch

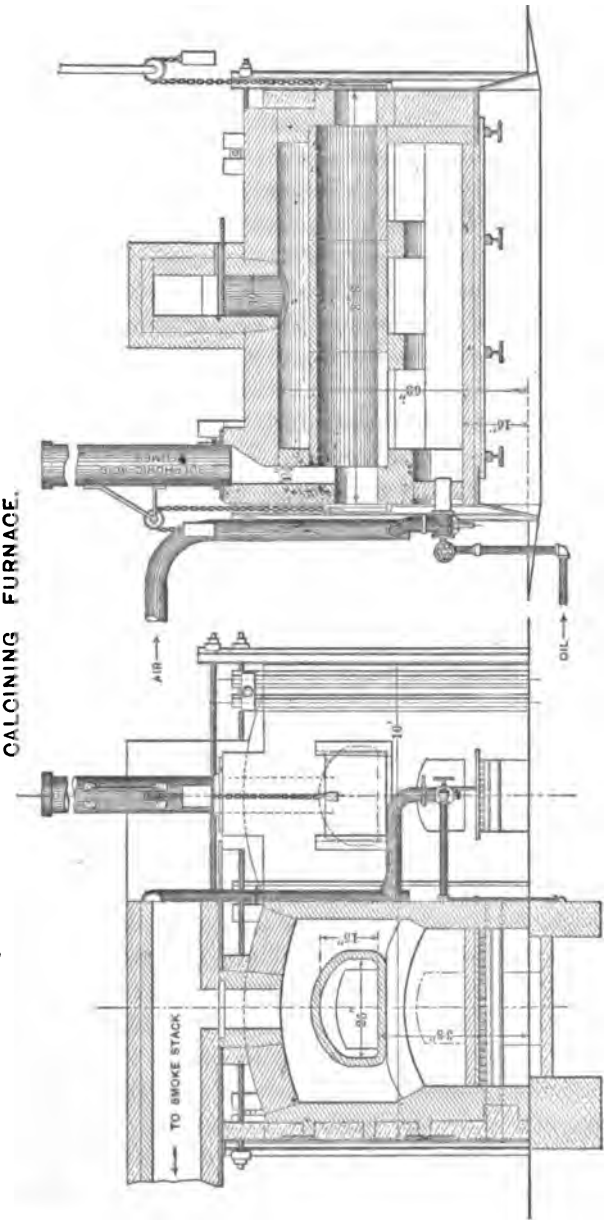


FIG. 3.
CALCINING FURNACE.

screw-conveyors. The upper flight carries away from the pulverizer,

the lower line returns to it. At intervals of 2 feet are gates, which, when open, put the upper flight of the conveyor in communication with the lower one. Suppose that a gate 10 feet away from the pulverizer is open; then all the slag that settles between this gate and the pulverizer is dropped into the lower conveyor and returned for re-grinding, while that, which settles beyond the 10-foot gate, is carried on and delivered at the end of the chamber, whence it is run into a receiving-barrel, and is ready for use. Should now the product prove too coarse, this is corrected by successively opening the gates 12 feet, 14 feet, 16 feet, and so on, away from the pulverizer, until the proper grade is obtained.

This arrangement is working in the most satisfactory manner on slag. The settling can be adjusted also by means of a cone-pulley, varying the speed of the 40-inch exhaust-fan, and thus changing the rapidity of the air-current.

On reaching the fan, the air still holds suspended about 40 per cent. of the dust originally carried, and this is the very finest and most valuable. To recover this, the exhaust from the fan is blown into a large dust-proof room, where the air-current is completely arrested. This room may be made of any shape that will suit the building. In Fig. 2 it is shown, for the sake of clearness, as a circular canvas chamber, while in reality it is of irregular shape, conforming with the slope of the roof, and placed at the side of the settling-chamber. The form is immaterial. What is wanted is a large cubic capacity. No baffling-screens or centrifugal dust-arresters will clean the air as effectually as a dead calm and plenty of room.

In the plant illustrated, when adjusted to grind to 225-mesh size, the air passes through the settling chamber in about 25 seconds, while the same air remains in the dust-room 120 seconds, at the end of which time it is comparatively free from dust. Return-pipes, 12 inches in diameter, then lead the same air back to the bottom of the machine to carry off a new load of ground material. In the pulverizer and settling-chamber a vacuum of about 1 to 1½ inches of water, and in the dust-room a pressure of nearly 1 inch, are maintained. A revolving scraper at the bottom of the dust-room carries the collected dust, which amounts to about 320 pounds per hour, through a slot in the floor, and drops it into a receiving-barrel. This dust is so fine that it cannot be measured by screens, and is good enough for any high-grade paint. When mixed with linseed oil and ground on an ordinary burr mill, it makes up into a dark olive-green color, so neutral in its tint that an addition of from 3 to 15 per

cent. of coloring-matter will change it into a bright blue, yellow, drab, or glossy black.

While thus about 40 per cent. of the slag is directly available, the remaining 60 per cent., collected in the settling-chamber and ground to a fineness of about 225 mesh, is used for producing the durable shades of dark red known as the Pennsylvania, Lake Erie, New York Central, and other railroad standards.

For this purpose 1000 pounds of ground slag are mixed with from 170 to 200 pounds of 66° B. sulphuric acid (specific gravity, 1.83). The quantity of acid is governed by the shade desired in the finished product. The mixing is done first by hand and then in a Chaser mill. When thoroughly mixed the slag is put away in a bin to "sweat." The chemical reaction is manifested by a rise in temperature to from 240° to 250° Fahr.

The slag consists mainly of silicate of protoxide of iron, and the sulphuric acid naturally expels the weaker silicic acid, forming a sulphate of protoxide of iron. If roasted with admission of air, this sulphate is dissociated into free sesquioxide of iron and sulphurous acid, which latter is drawn off.

The sweating occupies about four days, and is accompanied by the escape of steam and a greenish discoloration of the slag. When reaction ceases, 500 pounds of the mass is charged into each retort of the calcining furnace, shown in Fig. 3. This furnace is fired with crude oil, air for combustion being supplied under pressure by a Baker blower. The retorts are heated to cherry-red, and the slag, while being roasted, is agitated so as to bring it in contact with the air necessary for rapid oxidation. At the end of from three to four hours the heavy dark slag has changed into a light, flaky, red mass that is somewhat pasty. When the litmus-paper test shows that the acid is entirely evaporated, the slag is cooled down and passed through a second Cyclone pulverizer similar to the one described. The only difference in the grinding is, that the whole product is now reduced fine enough for the highest grade of paint, thus leaving no second residue. The pulverizer reduces about 800 pounds of calcined slag per hour.

The pulverizing-plant at Boonton, which consists of two mills, one running on raw and one on calcined slag, has a capacity of from 10,000 to 12,000 pounds of paint-stock per day of ten hours.

This plant, though in some respects rudimentary, like all pioneer establishments, has in it all the requirements for successful operation, and its products are receiving the acknowledgment and approval of the paint-trade and of consumers.

The slag-paints are remarkable for their durability, body, gloss and covering capacity, and for the stubbornness with which they resist all chemical reaction. They are cheap in price and uniform in fineness as in composition; the raw material is available in great abundance and at a nominal cost; and, as their superiority and the simple methods of preparing them become better understood, there is no reason to doubt that they might be universally used, and might form a most valuable by-product of the iron industry.

MINING IN HONDURAS.

BY W. A. THACHER, LEADVILLE, COLO.

(Glen Summit Meeting, October, 1891.)

NEARLY three years of experience in Spanish Honduras has placed the writer in possession of many facts concerning its resources which may be of interest to the Institute.

According to the most trustworthy surveys, Honduras is bounded on the north and east by the Caribbean Sea; southeasterly by the river Segovia (or Wanks), which flows into the Caribbean at a point called Cabo Gracias a Dios (Cape Thanks be to God), long famous as the favorite hiding-place of the celebrated pirate, Morgan; southwardly by a corner of San Salvador, the Bay of Fonseca, and the Republic of Nicaragua; Guatemala and San Salvador form the western boundary. It is one of the five republics composing Central America, has a population of about 340,000 and an area of 150,000 square miles, lying between the 13th and 16th degrees of N. latitude, extending from north to south 200 miles, between the Atlantic and Pacific, and 400 miles from east to west, between Guatemala and Cape Gracias a Dios. It is composed of 13 States or Departments, under one common law. The principal resources are valuable woods, hides, rubber, tobacco, coffee, fruit, cattle, silver and gold. It is of the mineral resources of the country that I will speak, calling attention to the most noted mines, and their relative position to Tegucigalpa, the capital.

The introduction of foreign capital into the country dates back about eleven years, and to-day there are at least twenty-five well organized companies.

The New York, Honduras and Rosario Mining Company.—This company is incorporated under the laws of New York, with 150,000 shares, at $10 a share. The property is about seven leagues from Tegucigalpa, on the side of a high N. and S. range of mountains, about 7000 feet above sea-level, and about 1200 feet above, and two miles distant from, a small village of about 1000 inhabitants, called San Juancito. The mine is on a gold- and silver-bearing quartz vein, from 3 to 15 feet wide, running northwest and southeast, and dipping south. The foot-wall is mainly a volcanic greenstone; the hanging-wall principally siliceous slate. The walls are hard and require but little timbering. The gangue is quartz, more or less copper-stained. The richest part, with few exceptions, lies next to the hanging-wall. The ore assays about $40 to the ton, and the mine gives employment to about 200 workmen, who, except the foreman, drill-men and machinists, are natives. The ore is transported by means of a 4000-feet endless steel cable from the mine to the mill (which has thirty-five 800-lb. stamps), crushed by Dodge breakers, stamped, and run over copper plates. From there it goes to the pans, being amalgamated in the usual way, and the tailings are run over Frue vanners. The concentrates are then roasted in reverberatory furnaces and amalgamated in special pans. I understand that it is intended to put the concentrators directly behind the batteries. The mill is driven entirely by water-power obtained from the San Juan and Escobales rivers, the water being carried through a flume nearly 3000 feet in length, thence down the mountain-side in a 16-inch wrought-iron tube, and discharged on an 8-feet Pelton wheel. This company, the oldest established one in Honduras, is well equipped with all modern tools, having a complete machine-shop, a first-class saw-mill, foundry, and all necessary wood-working machinery.

The Los Angeles Mining and Smelting Company.—This company is incorporated under the laws of New York, with a capital of $1,500,000. The property consists of the Animas and California mines, about two miles from the town of Valle de Angeles (Valley of the Angels), and twenty miles from Tegucigalpa. The Animas mine is a flat or "blanket-vein," having a dip of from 9 to 20 degrees to the south. The vein is several feet wide, with a foot-wall of quartzite conglomerate. The pay-streak, or mineralized zone, varies in thickness from 1 to 5 feet. Prior to its purchase by the present company, it had been worked for many years by the natives, the ore having been extracted so as to leave a perfect system of pillars

to support the roof, instead of timbering. The mine is very well developed, having several thousand tons of ore in sight, and the vein is proved for 700 feet in length. The ore is argentiferous galena, carrying a large percentage of zinc-blende and pyrite. The assay-average is about $60 in silver and 30 per cent. lead. The reduction-works are at the head of the Valle de Angeles, about a mile and a half below the mines. Here the Company has laid out a very pretty town-site, and built comfortable houses for the men. The smelter consists of two 15-ton water-jacket blast-furnaces, one large cupel-furnace, three Baker blowers, etc. The works, which are in charge of Mr. N. A. Foss, are among the few that have been running continually and successfully. Recently they have been selling their bullion to the Government Mint of Honduras, and as it contains no gold, they realize as much from it as they would by shipping it to the United States.

The Opoteca Mine.—The ore-deposits of Opoteca are no doubt among the most important mineral resources of Honduras, and the work done by the old Spaniards shows that immense quantities of ore have been extracted from them. Owing to the downfall of the mining industry after the declaration of Central American independence, these mines were left idle up to 1887, when a concession was granted to private individuals. The location is in the Department of Comayagua, near the center of the Republic, 112 miles from the Pacific, 125 miles from the Atlantic Ocean, and nearly 75 miles N. W. from Tegucigalpa. Indications and records show that these mines have been worked for over 200 years. Probably hundreds of thousands of tons of ore have been extracted, milled and treated in arrastras, ruins of which are to be seen on many of the streams near the mines. It is impossible to ascertain the amount of gold and silver that has been extracted. It is said that the Spaniards obtained from the selected ore an average of from $40 to $50 per ton. Very large quantities of ore were rejected, and it is estimated that 200,000 tons of ore are on the old dumps, which will yield from $10 to $20 per ton.

The Opoteca mining district seems to comprise a deposit of ore extending a mile or more in length, and developed for more than a quarter of a mile in width, with a depth in the old mines of 100 feet and more. The country-rock is carbonate of lime, in which the ore fills spaces of thousands of cubic feet. It is estimated that there are several millions of tons of ore in sight. The value of the ore, as shown by many assays, is about $20 in silver and $1 per ton in

gold, per ton of 2000 lbs. It is rare that a piece of high-grade ore is found in any part of the mine. The only new work done until now is a tunnel driven 200 feet in limestone, to tap the ore-body from below. The ore mills freely, and contains sulphurets and native silver. The natural conditions are exceptionally good, as mining can be done principally by open-cuts, and reduction-works can be run by water. Opoteca is about 2500 feet above sea-level, and the extreme range of temperature is from 55° to 85° F. A wide, easy-grade wagon-road, from Tegucigalpa to Comayagua, and thence to Santa Barbara, passes within three miles, making communication with the surrounding country and affording good freighting-facilities. This road is restricted to a 12 per cent. grade, and is limited in rock-cuttings to 12 feet in width, but in the more open country to 5 yards. It will be continued, to make northern coast-connections.

Yuscaran.—This mining district comprises the mountains of Yuscaran and vicinity. It is situated 45 miles S. E. from Tegucigalpa, 90 miles from San Lorenzo, on the Pacific coast, and is accessible by a practicable wagon-road. I will give only a rough sketch of the three principal groups of mines in this vicinity.

The town of Yuscaran is situated on a spur of the Monserrat mountain, 3300 feet above sea-level, having the peaks of Raspa Cola immediately on the southwest, which rises abruptly to an elevation of 6000 feet; a descent of nine miles leads to the valley which drains this part of the country, 1500 feet below the level of the town. The mines lie principally on the north side of the mountain, old dumps being found at altitudes of 3000 feet to 6000 feet above sea-level. The veins, with slight variations, run N. of E. and S. of W., and are true fissures, carrying sulphides of silver, copper, zinc, lead, iron, etc., together with antimonial sulphide of silver (dark ruby), and native silver in quartz. The ores on the surface follow the general rule and are free-milling, but as depth is obtained become refractory, and require roasting before amalgamation. The country-rock is a variety of syenite, intersected by dykes of greenstone, porphyry and basalt.

Of the three mining companies operating here, the Monserrat Mining Co. owns a concession on the higher part of the mountain, containing a number of veins, several of which are within a few feet of each other, and converging generally as they enter the mountain. At an elevation of 4500 feet the Company has driven a tunnel about 1000 feet into the mountain on one of the veins, but has been work-

ing until recently at a great disadvantage, carrying the ore out on men's backs in *tenates* (leather bags). Yet by this means enough ore has been extracted from the veins to supply 20 stamps during the greater part of the last year.

The company has many veins which in old times yielded rich returns, but were abandoned on account of the manner of working, which was from the surface downward, so that at a certain depth the water could not be handled. In addition to this, there was difficulty in working the base ore, which has been overcome by modern methods. The mills are driven by water-power, of which there is an abundance.

Messrs. Zurcher Bros. have driven, some 1300 feet below the Monserrat property, a tunnel about 3700 feet in length, which enters the mountain from the north, and has cut sixteen or more veins, many of which are found to carry pay-ore. They have cut the vein of the Novia, an old mine, and find that large quantities of ore have been extracted from the upper workings. An air-shaft is being pushed from the head of the tunnel to connect with the workings above, and to give several hundred feet of stoping-ground. Meanwhile the company is proposing to double its capacity, and also to erect large electric motors. By driving the tunnel a few hundred feet more, the Monserrat veins can be cut some 1800 feet below the lowest works of that company. The ores of this property are similar in character to those of the Monserrat.

Southwest of the Monserrat is a higher peak, called the volcano of Guinope, and composed of trachyte, an overflow of which has encircled the Monserrat at an elevation of 3000 feet above sea-level, coming around from each side of the mountain, and entirely covering the permanent formation for many feet in depth. On the south side this is perhaps 1000 feet in width. Below it the syenitic rocks again appear, with their veins and dykes as above described.

In this locality, east of the trachyte, is the group of mines called Guayabillas (named after the wild guava plant, which grows here in great abundance). One of these veins was worked by an English company about half a century ago. Several old natives have told me that they had to shovel the gold-amalgam from before the rocks of the arrastras, to allow them to move. At that time, no doubt, the property was producing plentifully, as the records show that the neighborhood was very prosperous. These veins are fissures containing more or less manganese, silver and gold. Messrs. Hodding, Pilcher & Imboden, the present owners, are building a fine 20-stamp

mill (10 wet and 10 dry) on the property, with revolving dryer, Howell roaster and Frue vanners.

The Guadalupe, Honduras, Gold and Silver Mining Co., Ltd.—This company was organized in London, with a capital stock of $650,000. The property is in the Department of Choluteca, District of Aramicina, near a town of the same name, about 30 miles from the Pacific coast, and about 85 miles S. W. of Tegucigalpa. The vein now worked runs N. W. and S. E., and is from 4 to 6 feet in width, the ore being quartz, with silver sulphides, carrying more or less gold. A 15-stamp amalgamation and concentration silver-mill is in course of construction, the work being pushed with a force of about 75 men.

Guasucaran Mine.—This celebrated silver-mine, one of the oldest in Honduras, situated about 60 miles from the Pacific and 30 miles S. W. of Tegucigalpa, at an altitude of about 5000 feet, has been kept in almost constant operation from its first working up to the present time, and has been continuously productive. Its history dates from the times of the early Spanish conquerors. Old documents, preserved in the Government archives, show that it was discovered in the early part of the 16th century. A party of Spaniards, travelling from the interior to the coast, and losing their way while crossing the steep mountain of Guasucaran, discovered among the ashes of their camp-fire small globules of silver. Returning to Spain at the first opportunity, they obtained a patent to work the vein by means of slave-labor. On the proclamation of independence, in 1821, the then owner, Antonio Rosa, exiled himself, and the mine was left to the natives, who occasionally visited it to extract ore from the rich portions of the vein. In 1850 and 1857 the mine was worked by Capt. F. Moore, who paid $60,000 for the property. Mr. John Connor, who came into possession in 1860, and until eighteen months ago was the entire owner, has been working it with good success ever since, although his milling-plant was most inadequate, consisting of an arrastra and a 5-stamp mill, with barrels for amalgamation.

The ore-bodies of the Guasucaran mine are found in fissures and contact-veins, as follows: One vertical vein, running through the center of the mountain, while a second vein, of nearly horizontal dip, intersects the vertical lode, forming large ore-deposits, rich in silver. The vertical vein is from 3 to 10 feet wide, its gangue consisting of quartz and barite, containing silver glance and zinc-blende. The country-rock is highly siliceous trachyte. The Guasucaran

and California Mining and Milling Co., formed in San Francisco about a year and a half ago, is just completing a 10-stamp lixiviation-plant, this process being considered best adapted for ores of this character.

San Bartolo Mines.—The San Bartolo grant, 3 by 6 miles in extent, is situated 55 miles N. of San Lorenzo and 50 miles S. W. of Tegucigalpa. This property was unknown to the Spaniards. It was discovered in 1849 by Don Cenecio Andino, by whom it was continually worked until 1883. In 1886 it fell into the hands of American mining men, but has not been worked since that time. The vein, which is from 2 to 5 feet wide, runs E. and W. along the mountain side, with a dip to the north, and outcrops 600 to 800 feet above the bed of a small river flowing parallel with the vein. The ore is a free-milling quartz, carrying black sulphurets of silver and chlorides, and yielding from $40 to $100 per ton.

I understand that during the time of Señor Andino over $300,000 was extracted, even with his crude way of working the mine. The same grant covers the El Plomo group of mines, largely worked 100 years and more ago.

Mines of San Andres.—This group is situated on the San Andres mountain, N. W. of Tegucigalpa, in the Department of Copan, about 25 miles S. W. of Santa Rosa, and nearly 50 miles N. E. of the line of San Salvador.

The vein-matter is a decomposed milky-white quartz, which in many places carries a large percentage of oxide of iron. The formation of the hanging-wall is white porphyry and micaceous slate. The foot-wall shows a gray slate, yellow porphyry, and in places syenite or schist. The trend of the vein is N. W. and S. E., with a dip to the south. This is another of the old Spanish mines, and, as the records show, was worked 100 years ago. The work that has been done is enormous. No recent visitor has yet thoroughly explored the underground workings, though the property has been examined lately by three different experts, who seem to agree very closely on the quantity and quality of ore, and pronounce it free-milling, with a yield of about $14 per ton of 2000 pounds. They report that two rivers, which unite at the foot of the mountains, will furnish jointly about 1000 miner's inches of water the year round.

The Suyape Silver Concession, Ltd.—This company was recently organized in London, with a capital stock of $500,000, to work a concession which contains a group of mines about 10 miles from Tegucigalpa. The Guadalupe, one of the mines in the group, has a

N. W. and S. E. course, and what is known as the San Luis lode crosses the Guadalupe at right angles on the east side of the mountain. The country-rock in which these mines occur, is a conglomerate marl. The Guadalupe has been opened by sinking a 35-foot shaft, from which a drift has been run about 70 feet on the vein. The vein-matter is a white conglomerate, impregnated with mineral, and the ore is found both in the quartz and in the cementing matter around it. The San Luis has been opened by sinking a 75-foot shaft about 300 feet from where it crosses the Guadalupe. The ore is about 4 feet in width, and of similar character to that above mentioned. The ore of both mines is argentiferous galena with zinc-blende, and will run about 30 ounces in silver and 12 per cent. lead. The company is working a small force, and proposes to develop the property thoroughly before purchasing any machinery.

The Dos Hermanos Mining and Milling Company.—This property is situated near a small village called San Juan, in the district of Corpus, in the department of Choluteca, and nearly due south from Tegucigalpa. This location gives it a great advantage over most others in the republic for the introduction of supplies, etc. It is about 25 miles from a shipping-point called Padregal, on the Choluteca river, which stream is navigable for light-draft vessels up to that point, even in the dry season. The Dos Hermanos mountains, from which the mine derives its name, stand nearly in the center of the concession, and are known to contain lodes and chimneys of gold- and silver-bearing quartz. The ore is a reddish iron-quartz, and is said to average $15 in gold and $5 in silver per ton. Some experiments have been made, showing that about 65 per cent of the gold can be extracted by free milling, and the tailings concentrated up to 95 per cent. of their assay value. Systematic mining is being diligently pushed, and a 40-stamp mill is soon to to be erected.

The Wermouth Concession was granted in 1888. It is situated in the mining district of Minas de Oro, in the department of Comayagua, about 80 miles N. E. from Tegucigalpa. The zone contains quartz veins and deposits of auriferous gravel; the latter in large quantities. The principal vein, the Eureka, has been opened by five levels, and the pay-streak varies from 6 inches to 3 feet. It is found to be very pockety, and will assay about $10 per ton. The country-rock is granite on the foot-wall and calcspar on the hanging; course N. E. and S. W., and dip N. W. about 68°.

The ore is principally decomposed quartz, carrying a large quantity of oxide of iron. The gold is entirely free, and easily saved by

ordinary milling. The gravel-deposits are now being prospected, and it is proposed to turn the river out of its old channel. The reduction-works consist of a large arrastra, driven by water-power, and capable of reducing from 10 to 15 tons of ore per day of 24 hours.

La Victoria Mining and Milling Company.—This concession is in the district of Santa Lucia, in the department of Tegucigalpa, and about 12 miles nearly east of the capital, the principal mines being the Victoria, Zopilotiera, and Santa Elena. The Zopilotiera vein is from 3 to 4 feet wide, striking east and west, and dips north from 5 to 10 degrees. The pay-streak, of black decomposed sulphuret of silver, is generally found on the foot-wall. The work done by the present company on this vein aggregates about 275 feet. The Victoria vein is a well-defined fissure, running east and west, and dipping about 20 degrees north. The ore and gangue are soft and much decomposed, and the silver-bearing minerals are arsenical and iron pyrites with ruby silver. On the Santa Elena, considerable work had been done by the former owners, who had proved the existence of chimneys and irregular bodies of good ore.

The topography of this zone is such that, in nearly every case, mining can be carried on above the drainage-levels, the latter being from 1200 to 1500 feet below the highest outcrop of the veins. A mill which was to contain some standard pulverizers, and other machinery, was not adapted to the treatment of these ores, and work has been stopped for an indefinite period.

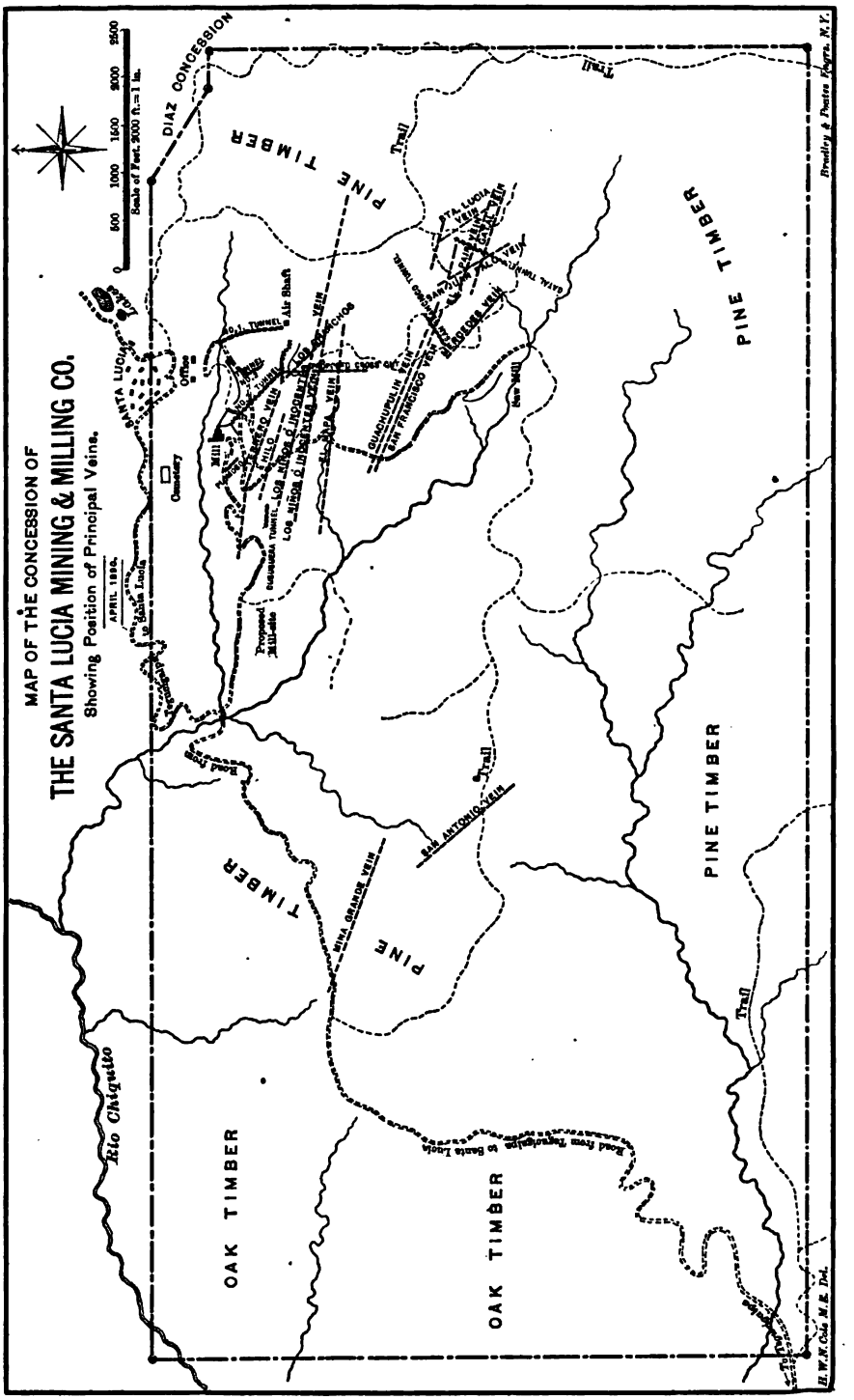
The New Orleans and Curaren Mining Works are located in the department of Tegucigalpa, about 75 miles from the city, nearly 45 miles from the port of Acetuna on the Pacific coast, and nearly 30 miles from the nearest telegraph station and post-office. The property comprises two claims, La Victorina and Dolores, together with two *pertenencias* or extensions. It has been worked continually for four years, and is fairly well-developed, its owners having driven about 7000 feet of drifts and tunnels. They have encountered a small fissure-vein from 6 inches to 5 feet in width, well-defined, carrying antimonial silver-ore, with more or less gold, and averaging in value about $35 per ton. The country-rock is porphyritic and syenitic. The works have a 10-stamp, free-milling silver-mill, with Frue vanner.

The Cuyal Silver-Mine, located on what is known as the South coast, is only a short distance from salt water, and about 90 miles from Tegucigalpa, near a small village called Langue. The mine has been worked in a very superficial and crude manner by natives

and prospectors, who have extracted hundreds of tons of ore, which was reduced in small *chimbos* (smelting-furnaces), this being smelting-ore, running from 50 to 70 per cent, in lead, and $100 to $200 in silver per ton. I understand from Mr. Connor, the present owner, that he made one shipment of 14 tons to Swansea which yielded $3000. The mine has produced about 250,000 ounces of silver.

San Marcos.—This grant, one league square, is situated on the road from the capital to the Pacific, near the town of Sabana-Grande, being about 50 miles from the port of San Lorenzo. The principal vein is a fissure, nearly vertical, in porphyry walls, and from 2 to 5 feet thick. It has been worked for nearly 3000 feet, and to a depth of 100 feet to water. The working of this mine was commenced in the year 1579, or 58 years after the occupation of Central America by the Spaniards. The ore is a free-milling quartz, carrying sulphuret of silver with some gold. A 10-stamp silver-mill was erected in 1877–8, and produced about $100,000 until the end of 1889, when the mine filled with water, owing to lack of pumping-facilities, and has since been idle. The battery-assays during this period averaged over $33 per ton, and the ore-body continues to descend. The deepest shaft has a depth of 240 feet.

Santa Lucia.—These mines, located about 9 miles east of Tegucigalpa, 4500 feet above sea-level, were among the first discovered in Honduras. All the veins are said to be fissures, striking east and west, and dipping about 70° north. The old workings show that the Spaniards did most on Mina Grande, at the western end of the concession. This mine was extensively worked up to the year 1821, at which time a depth of 450 feet had been attained, and the mine was abandoned for the want of sufficient ventilation. It is said that these mines paid over $1,500,000 in taxes (or what was known as the "king's fifth") to the crown of Spain, in the reign of Philip III. In 1886 this property fell into the hands of Americans; and in 1888 the Santa Lucia Mining and Milling Company was organized under the laws of New York, with a capital of $2,500,000. A concession was secured, 1½ miles wide from north to south, and 3 miles long from east to west, as shown by the accompanying map. A mill was erected, but unfortunately too far up the mountain for convenience. I understand from the manager, Mr. J. D. Stevens, that he has proposed to his company to move the mill to a better site about 700 feet further down the mountain, on a stream called Rio Chiquito. Nearly all the veins on the property would be accessible to a mill so located. I believe it is


MAP OF THE CONCESSION OF
THE SANTA LUCIA MINING & MILLING CO.
Showing Position of Principal Veins.
APRIL 1892.
DIAZ CONCESSION
Scale of Feet. 2000 ft.=1 in.
0 500 1000 1500 2000 2500
PINE TIMBER
OAK TIMBER
Trail
Rio Chiquito
Road from Tegucigalpa to Santa Lucia
Santa Lucia
Office
Cemetery
Mill
New Mill
Proposed Mill site
Dwelling House
Store
Powder House
Engine House
Air Shaft
Tunnel
Level
Stope
Vein
Los Angeles Vein
Los Angeles Tunnel
Los Angeles Stope
Los Angeles Air Shaft
Los Angeles Engine House
Los Angeles Level
Los Angeles Vein
Guadalupe Vein
San Francisco Vein
Santa Lucia Vein
Santa Lucia Tunnel
Santa Lucia Air Shaft
Santa Lucia Engine House
Santa Lucia Level
Santa Lucia Stope
Santa Lucia Vein
Blanche Vein
Blanche Tunnel
Blanche Air Shaft
Blanche Engine House
Blanche Level
Blanche Stope
Blanche Vein
Gale Vein
Gale Tunnel
Gale Air Shaft
Gale Engine House
Gale Level
Gale Stope
Gale Vein
San Antonio Vein
Mina Grande Vein
Bradley & Pastor Engrs. N.Y.
H.W.K. Del. N.E. Del.

intended to utilize the water-power of Rio Chiquito for running the plant and for lighting the works and mine by electricity. During the time required to remove the plant, the mine will be opened five or six hundred feet below the present workings, which will give large blocks of ground for over-hand stoping, and, according to present appearances, provide a sufficient quantity of ore for a long time to come.

Of the twenty-six known veins on the property, those with which the company is most familiar, and where most of the work for the past two years has been done, are the Ternero, Los Chanchos, Santa Ana, and Blanco. These have all been extensively worked at levels above the present elevation of the mill. Further up the mountain occur the Guachupulin, San Francisco, Mercedes, El Pais, Gatal, San Juan Palo, and Santa Lucia mines; all of which seem to lie parallel to the large veins below.

A new cross-cut tunnel, over 700 feet long, at the mill-level, has recently been completed. It cuts one of the large veins. Over 300 feet of drifting has been done on this vein, and large bodies of high grade ore are exposed, ready for stoping. They carry iron pyrites, black sulphurets, brittle and ruby silver, and a trace of gold. About 100 feet in the drift from the head of the long cross-cut tunnel, is another drift across the formation, the face of which is now in vein-matter; and it is expected to strike another one of the large veins known to exist in the levels above. No stoping has been done on the lower levels; and Mr. Stevens informs me that it is his intention to develop the ground thoroughly before extracting any ore, except from the headings, while driving on the veins.

There is a good saw-mill on the property, and a 10-stamp silver-mill, with a White mechanical furnace built by the Pacific Iron Works of San Francisco. The mill was originally designed to work one battery dry and one wet, but it was found that the loss was too great, and that the ores were not adapted to free-milling. The tailings are now concentrated and roasted in a reverberatory furnace, and elevated back to the mill, where they are treated separately.

Olancho Exploration Company, Ltd.—This concession, granted by the government to Major E. A. Burke, is, without doubt, one of the largest tracts of mineral land ever conveyed to one individual. It is in the department of Olancho, north-east of Tegucigalpa, and is divided into four sections, which have a net area of 715.6 square miles.

Section 1 contains 127.8 square miles, and contains the following concessions granted to other parties:

Retiro Honduras Gold Mining Company, Ltd., 1¼ square leagues; Honduras Gold Company, ¼ square league; Olancho Syndicate, 1½ square leagues; Cockerton Concession, 3 square leagues; Bell and Rector, ½ square league; William Harding, ½ square league.

The Retiro company has erected a 10-stamp mill, and opened a number of valuable veins which average well in gold. The Honduras Gold Company constructed a dam and flume in the Guayape river, but before the bar could be cross-cut and thoroughly tested a *cresciente*, or flood, carried away the works. The center of the bar, however, yielded about 65 cents per cubic yard. Bell and Rector have erected a 10-stamp mill, and are carrying on development-work from the proceeds of the mine.

Sections 2 and 3 have areas, respectively, of 174.8 and 195 square miles. The only location which has been made on Section 2 is known as the Trewenberg discovery, where a tunnel has been driven about 75 feet, and the face shows a vein about 4 feet wide, assaying well in gold.

Section 3 is, comparatively speaking, virgin ground, though many veins outcrop on the surface. No work has been done, except on the placers along the principal streams.

Section 4 covers 218 square miles. The Concordia Gold Mining Co., which has a concession 1½ miles wide and 3 miles long, built a 5-stamp mill, but there was not sufficient water for power and milling, and the works have been temporarily shut down until steam-power can be put in. Considerable work has been done on the Pizano and La Chabela concessions, and with comparatively good results. Perhaps the most interesting portion of this section is in the Guaymaca mountains, just east of the western boundary line. Here are to be seen very extensive workings, said to be over 100 years old. The oldest people in the vicinity know nothing of the operation of the furnaces, the ruins of which still exist. The mountains are bold, and the streams will furnish an abundance of power for driving heavy machinery.

Coal.—Occasionally float-coal has been found, and there are many localities throughout Honduras where, among the stratified marly shales, carboniferous seams occur, from which, without doubt, the above-mentioned float was derived, but most of the specimens contain about 75 per cent. of ash. A great portion of the stratified formations of this country, however, belongs to the Permian group, which is known frequently to contain workable coal-seams in other regions. Hence, it is not unreasonable to hope that a good quality of coal may yet be discovered in Honduras. Early in 1889 a con-

cession was granted to Mr. E. E. Chubbuck, of New Orleans, who failed to comply with the requirements of the mining law, in consequence of which his titles were forfeited. Later, in the spring of 1890, a coal-concession was granted to Mr. McCave, who also failed in his attempt; and in the summer of 1890 a third concession was granted to the energetic Mr. W. S. Valentine.

Oil, Coal and Salt-Boring Concession.—This was granted July 31, 1890, to Mr. W. S. Valentine, giving exclusive right for 20 years, throughout the State, to bore for salt, petroleum, gas, coal, and all precious metals encountered. Work was started on Oct. 4, 1890, near the capital, at Palmyra. At a depth of 118 feet, potable water was encountered and continued for 20 feet, at which depth it was found to be warm. The well was then encased with a 6-inch wrought-iron tubing, in order to shut out the water, and drilling continued. Strange to say, for 500 feet nothing was passed through but red marl; then some 30 feet of granite and quartz was passed; then some 20 feet of white sand; after which a red marl was again encountered, which continued for some 80 feet, and upon analysis was found to contain an abundance of alkali. The present depth is about 1100 feet, of which the last 500 has been the same red marl, sometimes hard and sometimes soft. Work is rapidly pushed, to reach sea-level, some 3200 feet in all, when it is supposed that salt, brine or oil may be struck. The well continues perfectly dry.

Telegraph and Telephone Lines.—In speaking of the mining industries of Honduras, I should not overlook a most important factor, which is the telephone, telegraph and mail service. We are indebted for this to Mr. B. Cecil, the enterprising manager, who in 1888 was made Director General of the service. In 1889 the government conferred upon him the honor of appointment as Postmaster General. He has shown untiring energy in this, as in the former position, by establishing a thorough system of couriers throughout the entire Republic. In reference to the telegraph, Mr. Cecil says: "We have now 2140 miles in use and 593 miles in course of construction, all of which is divided into 13 sections, with a main office in Tegucigalpa. There are 4 coast offices, Puerto Cortez, Truxillo and La Seiba on the Atlantic, and Amapala on the Pacific. In all there are 94 offices, with 324 employees, operators and clerks. In the year 1890 there were sent 260,003 messages, containing 5,741,768 words, valued at $174,550.50; messages sent to other republics, 8825, containing 144,973 words; messages received from other republics, 8517, containing 159,639 words; number of cables sent via La

Libertad, San Salvador, which connects with the C. and S. American Telegraph and Cable Co., 1275, containing 14,750 words, valued at $20,208.38. Value of plant, $444,879."

Tegucigalpa is well connected by telephone, there being outside lines to San Juancito mine, and to the Departments of La Paz, Comayagua and Santa Barbara, the last, by the way, being the longest telephone line in Central America—171 miles without relay.

Mail Service.—In this department there are 58 post-offices, employing 79 postmasters and clerks, and the service gives constant employment to 267 couriers, who, on their journeys over the mountains, carry 37½ lbs. of mail-matter on their backs. For the year ending 1890, the following mail-matter was handled:

	Pieces.
Received from exterior, ...	271,534
Sent from interior, ...	86,756
Handled in the interior, ...	662,756
Making a grand total handled in the Republic of, . . .	1,021,046

Tegucigalpa, which has been the capital of Honduras since the removal of the seat of government from the ancient city of Comayagua, is situated in the south-central part of the Republic, in the Department of Tegucigalpa, on the foot-hills of the main cordilleras, on the Rio Grande, at an elevation of 3200 feet, and is in W. Long. 87° 10′, N. Lat. 14° 15′, being about 90 miles nearly north of Amapala, the principal port on the Pacific, and about 300 miles by road from Puerto Cortez, the main Atlantic port. The delightful climate, which naturally follows the altitude of the place, makes Tegucigalpa, with its various beautiful parks scattered throughout the city, a most desirable residence. A fine system of water-works, recently completed by Messrs. Gibson & Cole, the American contractors, gives abundance of excellent water. The pipe-line, for 9 miles, is of 12-inch pipe, having a fall of 838 feet, followed by ½ mile of 7-inch pipe, with a fall of 900 feet. Six ornamental fountains are placed in various public squares.

Contracts have been given to light the city by electricity, the streets to be illuminated by 2000-candle-power arc-lights, while the public buildings, hotels and dwellings are to be supplied with incandescent lights. A well organized and thoroughly drilled police force, under the command of Col. C. D. Beyer, of New Orleans, preserves the peace and order of the city. The public buildings comprise a School of Arts, a College, with chairs of Law, Medicine, Engineer-

ing, Natural Science, etc., two Seminaries and a college for boys, in which the modern languages are taught, the study of English being a favorite.

The Palace and Congressional Hall (Government Building), which was recently remodeled under the direction of His Excellency, the President General Luis Bogran, is a building worthy of note, an ornament to the city and an honor to the nation.

Mining in Honduras has had the same vicissitudes as in the United States; but it is a notable fact that not one of the mines has proved a failure. In the north and south of the Republic are the gold-mining districts, while the silver belt extends clear across the country, and is fully 50 miles wide. Without doubt, there is more mineral wealth in Honduras than in any other country of the same size in the world. I have seen much of the country; and speaking as a miner of considerable experience, I can say candidly that there is no richer field for mining operations than Honduras presents to-day, and there is no country that does more to invite and encourage miners. Under the wise and liberal policy maintained by President Bogran, capital has been secure in all its investments; full protection has been furnished to every reputable business enterprise and the gates of the country have been thrown wide open to every intelligent effort calculated to develop its resources. His influence and example have brought the people of the country into full accord with his intelligent and liberal policy. Americans have taken advantage of this disposition on the part of the President; there is a large amount of capital from the United States already invested in the mines of Honduras and other enterprises; and there is still ample opportunity for safe and remunerative investment.

THE FUEL-SUPPLY OF THE UNITED STATES.

A Sketch of the Progress of Twenty Years in the Economy of Production and Consumption.

BY JOHN BIRKINBINE, PHILADELPHIA, PA.

(Presidential Address at the Glen Summit Meeting, October, 1891.)

Four meetings of the Institute have been held in the anthracite coal-fields of Pennsylvania, and excursions into the district, in connection with meetings elsewhere, have familiarized our members with

the development of this unique region. But the progress of that development has been so marked that, in view of our assembling within a few miles of where, two decades ago, the Institute was organized, a brief *résumé* of the history of the anthracite coal-mining industry, and some thoughts upon the fuel-problems of 1871 and 1891 seem to be specially appropriate on this occasion.

Seventy years ago, the "burning stones" which we recognize as anthracite coal were first mined, but what has now become a magnificent industry grew with comparative slowness for two decades; and it was not until 1842 that 1,000,000 tons of anthracite coal per annum were produced.

The opening of this fuel-field, however, soon exerted a stimulus upon railway- and canal-construction, some of the earlier enterprises and numerous later additions having been projected and maintained to carry anthracite coal to points of consumption or to navigable waters. And this stimulus reacted upon the industry itself. Larger shipments were demanded to meet the requirements of the market developed by these improvements.

From 1842 to 1864 the annual production of this form of fuel increased tenfold, from 1,000,000 to 10,000,000 tons, and this amount was again doubled (to 20,000,000 tons) in 1873. The shipments of anthracite coal for 1890 show another step in geometrical progression; for, if to the 36,000,000 tons of anthracite shipped in that year we add the coal used at the mines, a production exceeding 40,000,000 tons per annum is accounted for.

The astonishing record of the mining industry within this limited field can be best appreciated by comparing it with that of another branch of mining. The weight of anthracite coal mined in Pennsylvania in 1890 was nearly three times as great as the total iron-ore product of the United States; and this coal was obtained at greater average depth, and under conditions generally less favorable, than are found at the iron-ore mines.

The total output of the anthracite regions of Pennsylvania for the past seventy years, approximates 750,000,000 tons. Of this, over 12 per cent. was obtained in the forty years between 1820 and 1860; 15 per cent. in the decade following; nearly 25 per cent. between 1870 and 1880; and for the past ten years the output has averaged about 35,000,000 tons annually, aggregating nearly 48 per cent. of the whole.

Of the total output of the anthracite coal-region as recorded, one half has been mined since 1879, and the period of twenty years

which marks the existence of the American Institute of Mining Engineers, covers a reported production of anthracite coal of nearly 540,000,000 tons, or 72 per cent. of the total output of the region.

Mr. J. H. Jones, special census agent, places the value at the mines of the anthracite coal product of Pennsylvania in 1889 at $65,721,578, demanding, at 362 regular establishments, the services of over 124,000 employees, who received in that year nearly $40,000,000 as wages. The capital invested in the coal-lands amounts to almost $162,000,000; but this valuation does not include undeveloped properties, or any surface improvements except such as are necessary for mining and preparing the coal. If to the above amount were added the coal-lands held in reserve, the numerous dwellings, and the enormous railroad facilities specially built and maintained to convey the product of the mines and breakers to trunk-lines, the valuation would be greatly augmented; for no part of Pennsylvania is so well supplied with railroads as the anthracite coal-fields, and few points in the United States are connected with as many competitive systems of transportation as the section which the Institute visits during this meeting.

Our *Transactions* demonstrate that much of the progress in mining methods and appliances by which the above results were accomplished is traceable to members of the Institute; and of their work we can truly say, "the half has not been told" in the printed papers forming part of our publications.

Mr. Joseph D. Weeks, Special Agent of the Division of Mines and Mining of the Eleventh Census, has supplied the following estimates of the consumption of natural gas in the United States in 1889:

	Cubic feet.
Iron- and steel-mills,	171,500,000,000
Glass-works,	18,750,000,000
Other industrial establishments,	236,900,000,000
Heating and cooking,	62,500,000,000
Pumping oil,	7,500,000,000
Drilling and operating oil- and gas-wells,	30,000,000,000
Other uses,	25,000,000,000
Total,	552,150,000,000

He gives as the approximate quantity of coal and wood replaced with natural gas 9,600,000 tons of coal and 69,000 cords of wood. Of this displacement, 71.5 per cent. is credited to Pennsylvania, 17.3 per cent. to Ohio, 7.5 per cent. to Indiana, and 1.4 per cent. to New

York; the balance being distributed among the other States. From these figures I have estimated the approximate value of the fuel thus replaced in 1890 at $20,000,000.

An approximate review of the consumption of all kinds of fuel for the past twenty years may be obtained from the following census data:

	1870.	1880.	1890.
Gross tons of anthracite coal,	13,925,229	25,580,189	40,714.721
Gross tons of bituminous coal,	15,356,619	38,242,641	85,383,059
Bushels of charcoal, . .	.....	74,008,972	90,000,000*
Cords of wood, . . .	.....	145,778,137	180,000,000*
Barrels of petroleum, . .	5,260,745	26,286,123	34,820,306
Natural gas, value in coal displaced, ...	No report.	No report.	$20,000,000*

The fuel consumed in the United States, exclusive of natural gas, but including the coal and wood converted into gas, requires the conveyance, by various methods, of nearly one and a half million gross tons each day of the year (no unimportant factor in the national problem of transportation), and demands the energies of over one million wage-earners to mine, cut, handle, and convey it to points of consumption. The apparent fuel-consumption, *per capita*, in the United States, is in calorific value equivalent to 3½ tons of coal per annum. Possible economies may reduce this to 2½ tons for the same amount of work performed.

By reason of the extended area of the bituminous coal-fields, their development has greatly benefited the entire country. This fuel has supplied most of the illuminating- and fuel-gas, and, in addition to its liberal use for domestic and steam-producing purposes, the coke resulting from its distillation has influenced the iron industry to a wonderful extent, as is shown by the following figures, taken from the reports of the ninth, tenth, and eleventh censuses:

	Gross tons.		
	1870.	1880.	1890.
Pig-iron, total product, . .	1,832,876	3,375,912	8,553,374
Pig-iron from coke or bit. coal,	510,000†	1,351,765	6,711,974
Manufactured iron, . . .	1,287,347	2,101,183	2,518,193†
Steel (*a*, finished product; *b*, ingots)	44,426*a*	1,022,956*b*	3,988,327*b*

Our total output of bituminous and anthracite coal in 1890 was

* Estimated.

† As the Census statistics had not been issued, the above figures were taken from the Report of the American Iron and Steel Association.

75 per cent. of that mined in Great Britain, the country which supplies nearly as much coal as all others in the world combined, exclusive of our own. In other words, the United States now produces about 30 per cent. of the world's output of coal, while it consumes, in addition, great quantities of wood, charcoal, oil, and natural gas. The United States is the great fuel-consumer of the world.

Over 2,000,000 cords of wood are annually cut to produce the 90,000,000 bushels of charcoal which I estimate were used in smelting ores of iron and of the precious metals, and for other specialties. The cord-wood used as fuel for various purposes largely exceeds that cut for charcoal, but the amount can only be roughly approximated. The total wood of all kinds used for fuel is estimated by Mr. B. E. Fernow, chief of the Forestry Division of the United States Department of Agriculture, at 180,000,000 cords per annum.

Peat is used to a moderate extent only; but various methods of drying or preparing it are passing through experimental stages.

In thirty-two years the product of petroleum has grown from 2000 barrels to 35,000,000 barrels annually, reaching a total for the whole period of 450,000,000 barrels. Of the present output, approximately 40 per cent. is used for fuel; and this has a calorific value, 3 barrels of petroleum being taken as equivalent to 1 ton of coal, of over 4,500,000 tons of coal annually. The convenience of application favors the use of petroleum in localities where coal commands relatively high prices, and in cases where intermittent firing is employed; and we may anticipate a growing demand for this form of fuel.

It is perhaps impossible to determine the quantity of natural gas used in this country, but a measure of its importance may be found in the equivalent money-value of coal it has displaced, which I have estimated above at $20,000,000 annually. Moreover, the extensive introduction of natural gas for industrial and domestic uses has stimulated the employment of producer-gas; and in many instances the manufactured gaseous has supplanted the natural solid fuel.

The magnitude of the quantities represented by our fuel-supply emphasizes the importance of economies in its use. A saving of one per cent. of the fuel of all kinds consumed in the United States would be equivalent to 2,300,000 tons of coal per annum. Each of our members will aver that, in almost every case of which he is cognizant, one per cent. of the fuel used could be readily saved, and in numerous instances the possible economies appear to be much greater. If all the fuels produced in the United States were used for one year

for the generation of steam, they would furnish continually, through boilers and engines of ordinary efficiency, nearly 12,000,000 horse-power; but if applied to boilers and engines of high economy, the resulting energy would be augmented to 25,000,000 or even 30,000,000 horse-power. An average reduction in American blast-furnaces of one hundred-weight of fuel per ton of pig-iron made would amount to an annual saving of nearly a half million tons of coal, and proportionate economies in iron and steel manufacturing processes would more than double this saving.

The crowds of coal-pickers covering a city's ash-heaps attest the waste of fuel imperfectly consumed for domestic and manufacturing purposes, and the "smoke nuisance" (the abatement of which is always in the future) is constantly before us as an indication of improvidence. The evil is to be overcome, if we believe the announcements of trade-circulars, by patented boiler-settings, which facilitate the evaporation of water at a rate in excess of theoretical perfection; by smoke-consuming devices, which effect such economies that it will pay to burn coal merely to utilize the smoke; or by stoves or furnaces which, through some secret of nature confided to the inventor only, utilize upward and downward draughts in the same chimney, the downward draught bringing the combustible myth to intensify immensely the heat of the normal fire. Economies are also claimed for special details in blast-furnaces, heating-furnaces, puddling-furnaces, or other metallurgical constructions, some promising a return of more heat-units than are accepted as theoretically attainable.

While many of the so-called fuel-saving inventions are farcical, because they are based upon unsound theory, we are indebted to others for marked economies in fuel-consumption, which have made it possible to advance metallurgy and manufactures to the position they now occupy. It requires but a brief retrospect to recognize the true economies, and for this we need note only improvements made during the existence of the American Institute of Mining Engineers.

Suppose for a moment that the quantity of pig-iron and manufactured iron and steel now demanded by the United States had to be produced, manufactured, manipulated, and transported by methods in use two decades ago—how many tons of fuel would be wasted in producing the 8,553,374 tons of pig-iron we made in 1890, if the blast-furnaces were equipped with open fronts and fore hearths, following the custom of charging more fuel than the furnace could consume, so as to shovel it out at the bottom? What additional fuel would be required if gas-producers were not connected with the

furnaces in our mills and steel-works, or if the direct methods of conveying molten metal from blast-furnace to converter, and hot ingots or billets from converter to mill, or from one mill to another, were abandoned, and if our mills returned to the former practice of allowing the metal to cool after each stage of treatment?

The developed coal-mines of the United States could not possibly supply the present demands of the country if our industries, our locomotives, and steam-vessels consumed fuel at the rate per ton of product or per horse-power that they did in 1871. Neither could the coal-mines, as opened and equipped in 1871, produce the fuel now demanded.

Among the economies already achieved none are more notable than those adopted in and about the mines, by which the coal is removed with less loss in mining, and prepared so that, of that which formerly went to the dump-piles, nearly all that is combustible is reclaimed. The application of finer sizes and even of the slack coal and dust to useful purposes is entitled to a prominent position among the fuel-economies.

A critical review of the advances made in the methods and processes of mining, preparing, handling, and transporting coal, of producing and using steam, of smelting, refining, and manufacturing metals, would probably show that we are now able to accomplish with one ton of combustible mined, results which, twenty years ago, would have required the mining of two tons or more. If then he is entitled to credit as a benefactor who causes two blades of grass to grow where one grew before, we have cause for congratulation and for just praise that, in 1891, one ton of combustible taken from the mines benefits the country as much as, or possibly more than, two tons as removed, prepared, and used in 1871.

Such results, however, are not due to one man; to accomplish them, thousands of active minds have worked in one general direction, against difficulties and discouragements. The geologist, the mineralogist, and the explorer have done their part in locating and developing additional sources of fuel-supply. The mining engineer, with the aid of the constructor and the mechanic, has brought to the surface, prepared, and delivered to consumers the solid, liquid, or gaseous fuels at low cost, while the laboratories of the chemists and physicists have contributed valuable data as to the composition and utilization of various fuels to assist the metallurgist and engineer in applying them. All of the classes mentioned are represented in the list of members of the American Institute of Mining Engi-

neers. Our roll of 2300 embraces in each department distinguished specialists, whose contributions to the *Transactions*, or work in practice, have aided in no small degree to accomplish what has been achieved. Yet, as indicated, there remains much to be accomplished, and the purpose of these remarks has been to invite attention, in a general way, to the magnitude of the fuel-problem, and, while recognizing the advances made in the past twenty years to encourage the endeavor after further economies in the utilization of fuel.

A NEW SYSTEM OF ORE-SAMPLING.

BY H. L. BRIDGMAN, BLUE ISLAND, ILL.

(Glen Summit Meeting, October, 1891.)

THE correct sampling of ores is a subject of far greater importance than is usually conceded to it. Of the little which has been published on this subject, the recent paper by Mr. Glenn, with the accompanying discussion by Dr. Raymond (*Trans.*, xx., 155), is the latest and best, and well worthy the careful perusal of all interested in accurate sampling, whether by hand or by machine.

It is doubtless true that thorough hand-sampling gives as good average results as can be obtained; but all hand-sampling is subject to the possible incompetency, carelessness, or dishonesty of the workmen employed. It is, moreover, tedious and expensive, requires large space, and usually entails considerable delay, both in unloading and in sampling. For these and other reasons, machine-sampling is now regarded with more favor than ever before.

So-called machine-sampling may be accomplished in two radically different ways:

1. Part of the stream of material is taken for the whole time.
2. The whole stream of material is taken for part of the time.

Under the first case the stream, as it issues from the delivery-spout of the crusher, is cut continuously in any given proportion, such as one-tenth, one-fourth, or one-half, by some fixed apparatus, which, as it contains no moving parts, is strictly speaking not a machine at all, but may be termed an appliance. The principle under which these appliances operate is now pretty generally conceded to be wrong, because of the practical impossibility of cutting such a

stream continuously, in such a way as to obtain a proper proportion not only of the stream itself, but particularly of its "coarse" and "fine" parts. The leading exponents of this principle are the sampling-shovel and the pipe- or spout-sampler. The latter, which is the best instrument of its class, has the disadvantage of being very difficult to inspect and to clean; it is still in use in some places, but has been abandoned in others.

Under the second case, the stream is cut in such a way that the entire flow is diverted at certain stated intervals, yielding a sample in which both "coarse" and "fine" of the original lot are fairly represented. The samplers which effect this are the machines proper, and the best-known examples are perhaps the Collom and the Brunton, both of which have done excellent work.

In all sampling operations, whether by hand or machine, certain fundamental truths are recognized, the most important of which is doubtless the proposition that the more uniform the material sampled the truer will be the sample. Such uniformity may be natural (as in the case of solutions and of certain ores and products) or it may (and usually must) be produced by crushing and mixing. The finer the former and the more thorough the latter, the better the result. Some materials, carrying "metallics," horn-silver, etc., cannot thus be rendered uniform, and are particularly difficult to sample.

It is theoretically correct to precede each quartering or cutting-down by a crushing, coarse at first and finer with each succeeding cut. In practice, however, rarely more than two crushings are employed, and these are moderately fine, no matter how many cuts may be made. Some works screen their material as it is crushed, returning the "coarse," until all has been reduced to a certain maximum size. This is notably true of hard iron-ores. The writer believes this practice to be entirely wrong. The result of the screening is a more or less decided separation of the material into its component parts, as, for instance, into quartz and iron oxide, thus giving, not the desired more thorough mixture, but a practical undoing of the natural mixture already existing. Of course, re-mixing may be resorted to; but it would be preferable to crush the entire material once as fine as possible or as may be desirable, and then to sample it, permitting nothing to come between the crushing and the sampling. This latter point is always important, and hence sampling-machines should be set directly below the crushing-plant, thus avoiding much unnecessary cleaning and risk of "salting" subsequent samples.

The assertion that very wet ores can only be sampled by hand is

an error. Anything that can be crushed can be sampled by a suitable machine, and material that is too wet to crush cannot be properly sampled, even by hand. Indeed, it is certain that crushing expense would be greatly reduced and better samples secured, whether by hand or machine, were the drying of original samples more frequently practiced than is at present the rule. The plant required is not particularly expensive, nor is much handling of the material necessary; while the heat to evaporate 10 per cent. of moisture from 100 tons of ore every 24 hours would be furnished by the exhaust-steam of a 60 or 70 horse-power engine.

Up to the present time no machine has been introduced which does more than the first portion of the work of sampling. It was the writer's desire to remedy this deficiency by devising something which would give practically finished samples; and the result of his efforts is the machine described in this paper.

Numerous designs were made, embodying the various possible combinations, and it is believed that the one adopted includes the best qualities of all; it is a rotary machine (as distinguished from an appliance) which takes the whole stream for part of the time, and which, in a single passage of the material through it:

1. Gives two or more absolutely independent samples (double samples).

2. Cuts down (quarters) each of these samples a sufficient number of times to give the smallest final samples desirable without re-crushing.

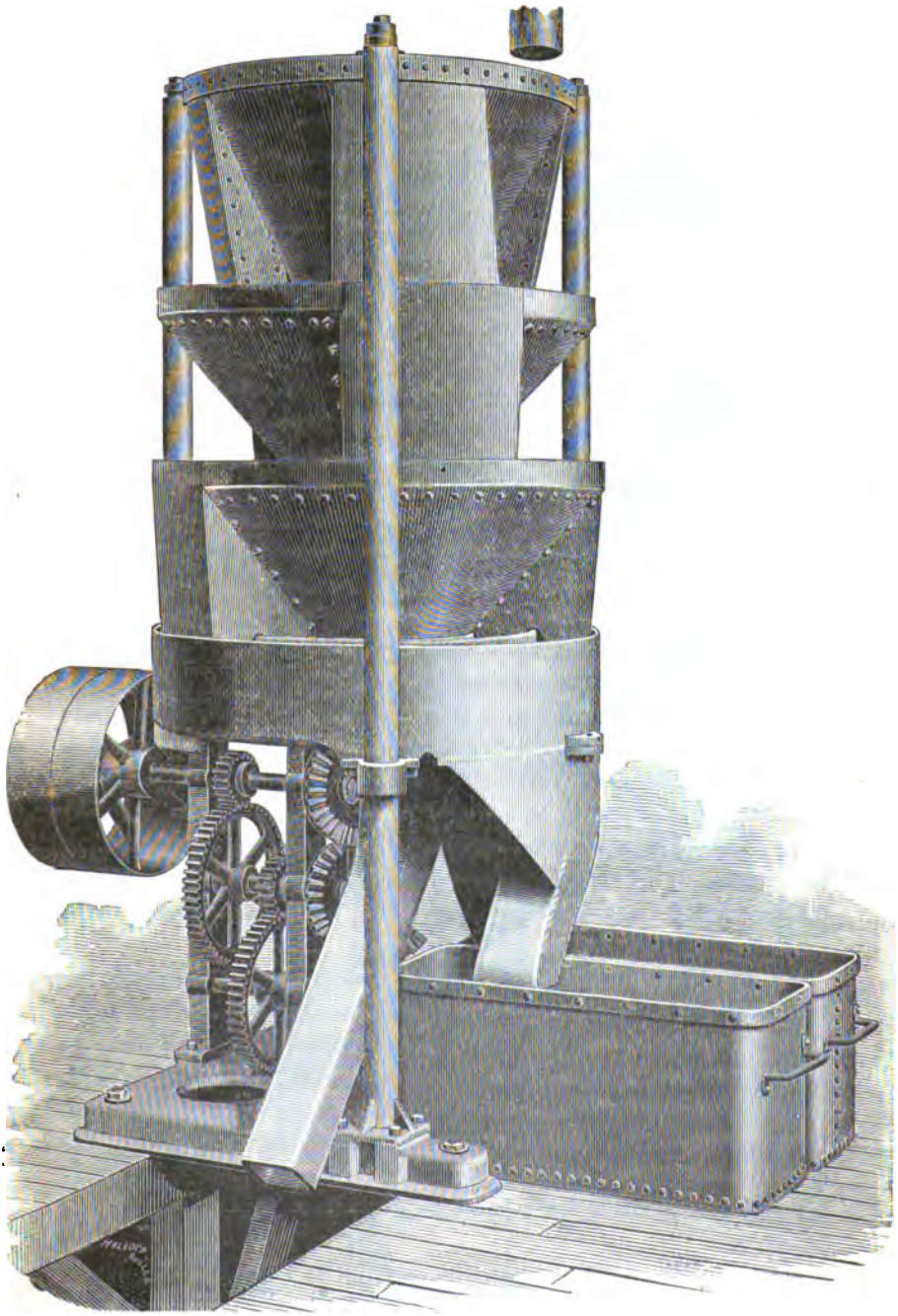
These two principal features are possessed by no other machine of this class, and are supplemented by the following hardly less important ones:

3. It is adjustable, to give different-sized samples, according to the grade and character of the material treated. The lower the grade, the more uniform the character, and the finer the size of the material, the smaller the final samples may be.

4. All the passages being large and straight, its capacity is very great (from 15 to 25 tons an hour), and it will sample any material that can be fed to it. Ore carrying 10 per cent. and over of moisture offers no difficulties; nor does the presence of strings, pieces of wood, and other extraneous substances affect the general result.

5. The quickness with which the samples are obtained makes it possible to use the machine-samples for the determination of the moisture, as well as of the metallic and other constituents, which is, of course, a decided advantage.

FIG. 1.



Mechanical Ore Sampler. Machine A. General View.

6. This quickness, moreover, makes it possible to sample all material as soon as received, with the consequent avoidance of demurrage on cars, and other expensive and annoying delays.

7. The machine gives good results under any ordinary working conditions; it takes its feed directly from crusher or rolls, regularly or irregularly, fast or slow, as the case may be; it requires no attention, except for cleaning out and for the removal of samples, and, in fact, it transfers the important function of sampling from the domain of watchful care and discretion to that of mere routine. Being perfectly impartial, and giving double samples, it removes ground for disputes, and renders "salting" practically impossible.

8. It requires less than one-tenth as much space as hand-sampling, and delivers the discarded part of the sample in such a way that it can be disposed of by elevator or similar means, requiring no hand-labor.

9. It is entirely self-contained, and very compact.

10. It may be carefully and fully inspected while at work, without the necessity for the near approach of any person, so that the most suspicious ore-seller may easily assure himself of the fairness of the operation.

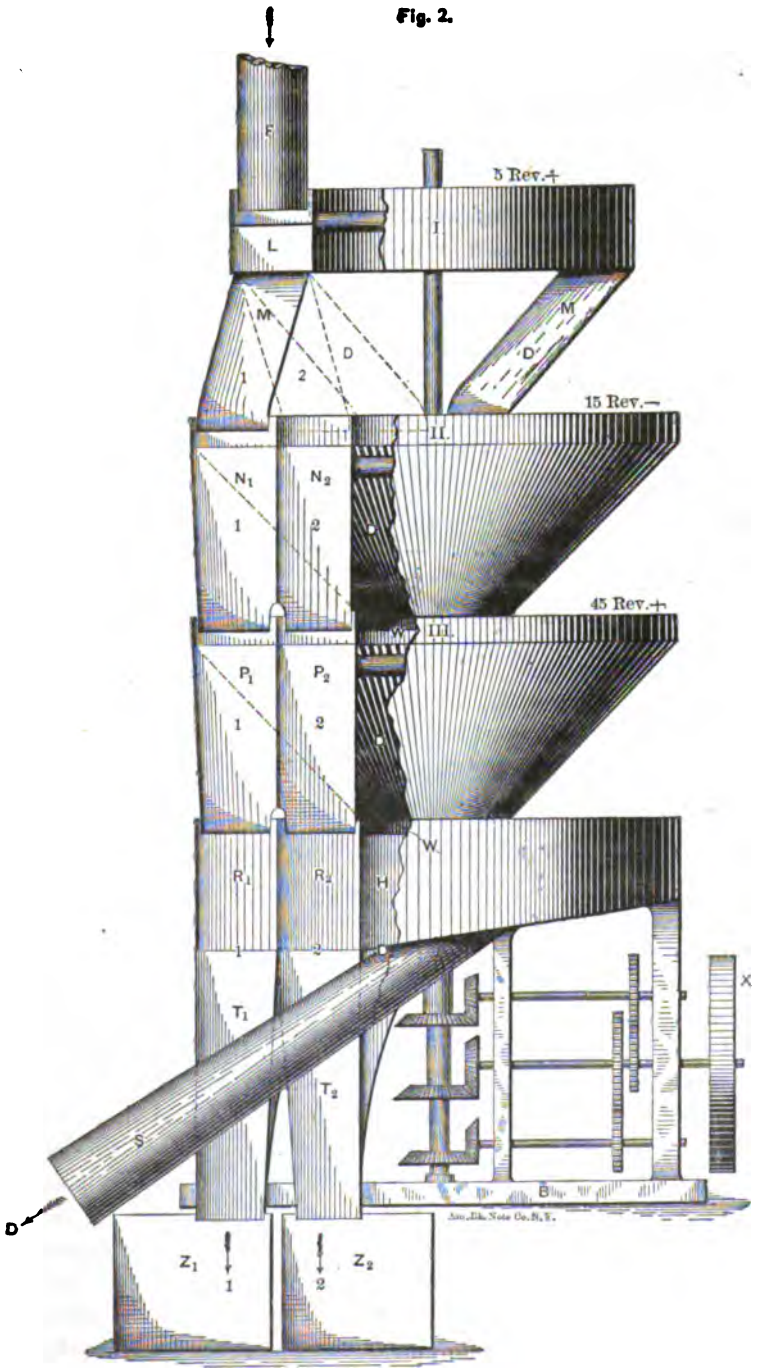
11. It may be thoroughly cleaned by one man in 15 minutes or less, and, if desirable, may be washed out with a water-hose.

12. It may be enclosed in a dust-case. This, however, is rarely necessary, as it is usually better slightly to moisten dry ores before crushing—these ores seldom requiring moisture-determinations. Should such moistening be undesirable, little harm can come of running the ore dry, as no men are about who can be injuriously affected, nor can the flying dust have any marked effect on the sample, and certainly not on any subsequent samples. Wet ores, of course, make little or no dust.

13. It is very inexpensive, both to operate and to maintain, the total cost of sampling with this machine being only about one-tenth that of hand-work. It is provided everywhere with ball-bearings and graphite bushings, which require practically no lubrication (with its accompanying dirt and annoyance) and make it possible to run the machine with much less power than may be continuously exerted by one man. All parts exposed to wear are heavy, while ample provision is made for repairs, duplicate parts being readily obtainable.

14. It is preferable to crush the material to 1 inch or less, but, if necessary, much coarser product may be run.

Fig. 2.



Mechanical Ore-Sampler. Size A. Total Height, including Sample-Buckets, 7 feet 6 inches.

The intention has been to provide not only a first-class sampling-machine, but also a system which should give the machine its greatest effect by embracing all the sampling-operations, from the receipt of the material to the filling of the assay-bottles. To this end, the following machines and apparatus have been designed to meet the various requirements:

Size A, Sampling-Machine.—This gives double samples and three cuts (quarterings) on each sample. It is intended for original samples of any size, and gives final samples of 1 to 2 per cent. of the original weights. Its average capacity is about 20 tons an hour.

Size B, Sampling-Machine.—This gives a single sample and three cuts. It is intended for the smaller original samples and for work requiring only one sample, as, for instance, iron-ores and the various furnace supplies and products. It is also intended for use in cutting down the crushed final samples of the A machine. Its average capacity is from 2 to 4 tons an hour.

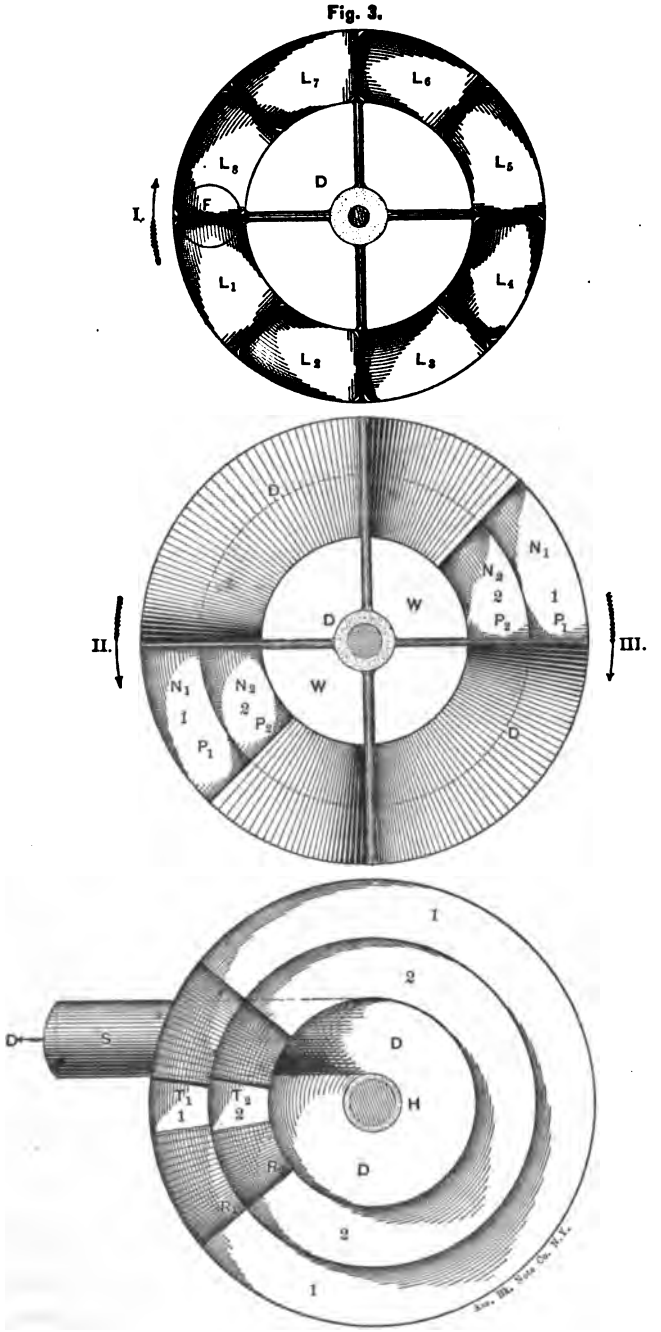
Size C, Sampling-Machine.—This gives laboratory samples for all small work.

Mixer and Divider.—This performs the final preparation and distribution of the assay-samples.

Machine A.

This machine occupies a floor-space of 3 by 4 feet, and has a total height of 7 feet 6 inches. It is self-contained, requiring only to be bolted to the floor and to have feed-, discharge-, and belt-connections made. Fig. 1 shows the machine as it is built, while Figs. 2 and 3 give the diagraphic sections and details, some minor changes and omissions having been made for the sake of clearness. The machine consists essentially of three apportioners, I, II, and III, all driven by the one pulley, X (usually tight and loose pulleys), and three stationary, concentric receptacles, R_1, R_2, and H, so constructed that any material falling into them will pass out through the spouts T_1 and T_2 into the sample-buckets Z_1 and Z_2 or through the spout S, which discharges the rejected portion of the sample. Apportioners I and III revolve in the same direction, apportioner II in the opposite direction; I at about 5, II at about 15, and III at about 45 revolutions a minute. That is to say, each apportioner moves actually three times as fast as the one above it, and in the contrary direction, or, relatively, four times as fast. By the use of this expedient of contrary revolution, the same relative speeds are obtained

Fig. 3.



Mechanical Ore-Sampler. Size A.

as though, all revolving in the same direction, the actual speeds were respectively 5, 25, and 125, at which latter speed centrifugal force would become very troublesome.

The upper apportioner, I, consists of two concentric rings, divided by 8 partitions into 8 equal topless and bottomless compartments, L, from each one of which leads an adjustable spout, either as M–1, or as M–2, or as M–D. Set in rotation, spout M–1 would describe a certain circular path, 1–1; spout M–2 a certain other path, 2–2, and spout M–D a third path, W (see Fig. 3).

The intermediate apportioner, II, is merely a conical funnel, having, besides the large outlet W, four vertical shoots, N_1–N_1 and N_2–N_2, through its sloping sides, as shown in Fig. 3; each one of these shoots forms one-eighth of the circular paths covered by the spouts M–1 and M–2 respectively.

The lower apportioner III is of the same construction as II and bears the same relation to it that II bears to I.

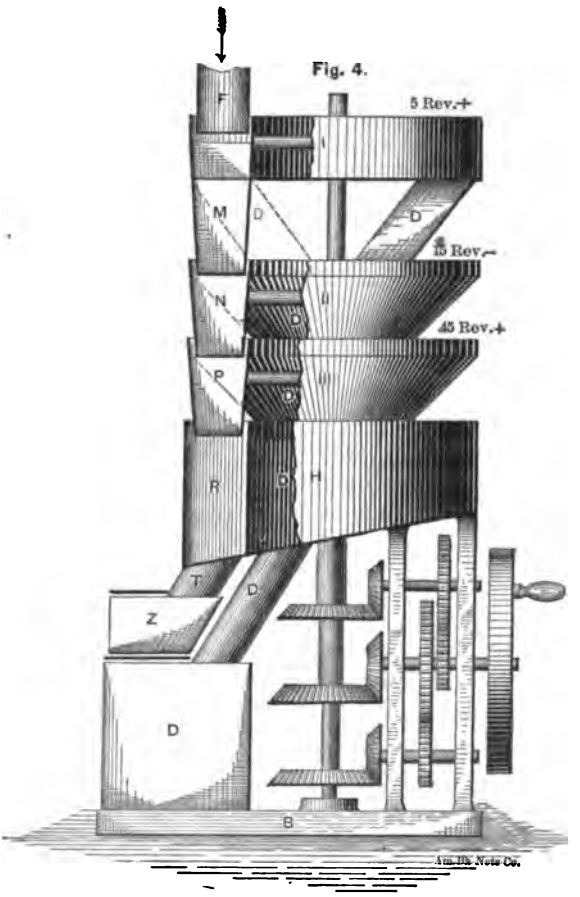
An example will best illustrate the operation of the machine. It may be assumed that an original sample of 40,960 pounds (the 960 being added to avoid fractions) is to be put through the machine; that the time required will be one hour; that the speed of the machine is such that the upper apportioner, I, will make 320 revolutions in that time, and finally that the ore is of such grade and character as to only require the smallest sample that the machine will give. Under these conditions, one of the spouts, M, would be set as M–1, one (the opposite one) as M–2 and the remaining six as M–D (Fig. 2).

The machine would then be set in motion, and the flow of material, previously crushed to below one-inch in size, would be started through the feed-spout, F. The series of operations which would follow may be best understood by reference to the upper part of Fig. 5. This diagram, like other similar ones accompanying this paper, is drawn approximately to scale, as a graphic representation of the work done by the machine under the conditions stated.

It is evident that at each revolution one 320th part of the whole lot, or 128 pounds, will pass through the feed-spout, F. Of this amount six-eighths or 96 pounds will be discarded by the six spouts, M–D, passing down through W, W, H and so through the spout S and out of the machine, while one-eighth of the 128 pounds, or 16 pounds, forming the first cut of the first or outer sample, will pass through the spout M–1 and the remaining one-eighth, or 16 pounds,

forming the first cut of the second or inner sample, through the spout, M–2.

These two first cuts will proceed side by side, by separate paths through the same series of operations, and whatever applies to the one, applies equally to the other; it will, therefore, suffice to follow



Mechanical Ore-Sampler. Size B. Total Height, 3 feet.

the first sample. This one-eighth or 16 pounds, having been cut from the mass by the partitions of the compartment L_1 of which M–1 forms an extension, will drop nearly vertically through M–1 on its way to the sample box, Z_1. As it leaves the spout, M–1, during the one-eighth of a revolution that is occupied by the said M–1 in passing beneath the feed-spout, F, it will be intercepted by the inter-

mediate apportioner, II, which in the same time will have made one half-revolution (relatively to I).

Since the vertical shoot, N–1, occupies one-fourth of the semi-circumference of II passing beneath the spout, M–1, it follows that one-quarter of the 16 pounds, or 4 pounds, will drop vertically through this shoot as the second cut of the first sample. The remaining three-quarters, or 12 pounds, will pass down the sloping sides of II and be discarded through W, W, H and S.

In precisely the same way the second cut of 4 pounds will be quartered by the lower apportioner, III, 3 pounds being discarded and 1 pound, as the third and final cut, passing through the vertical shoot P_1 and the spout T_1 into the sample bucket Z_1.

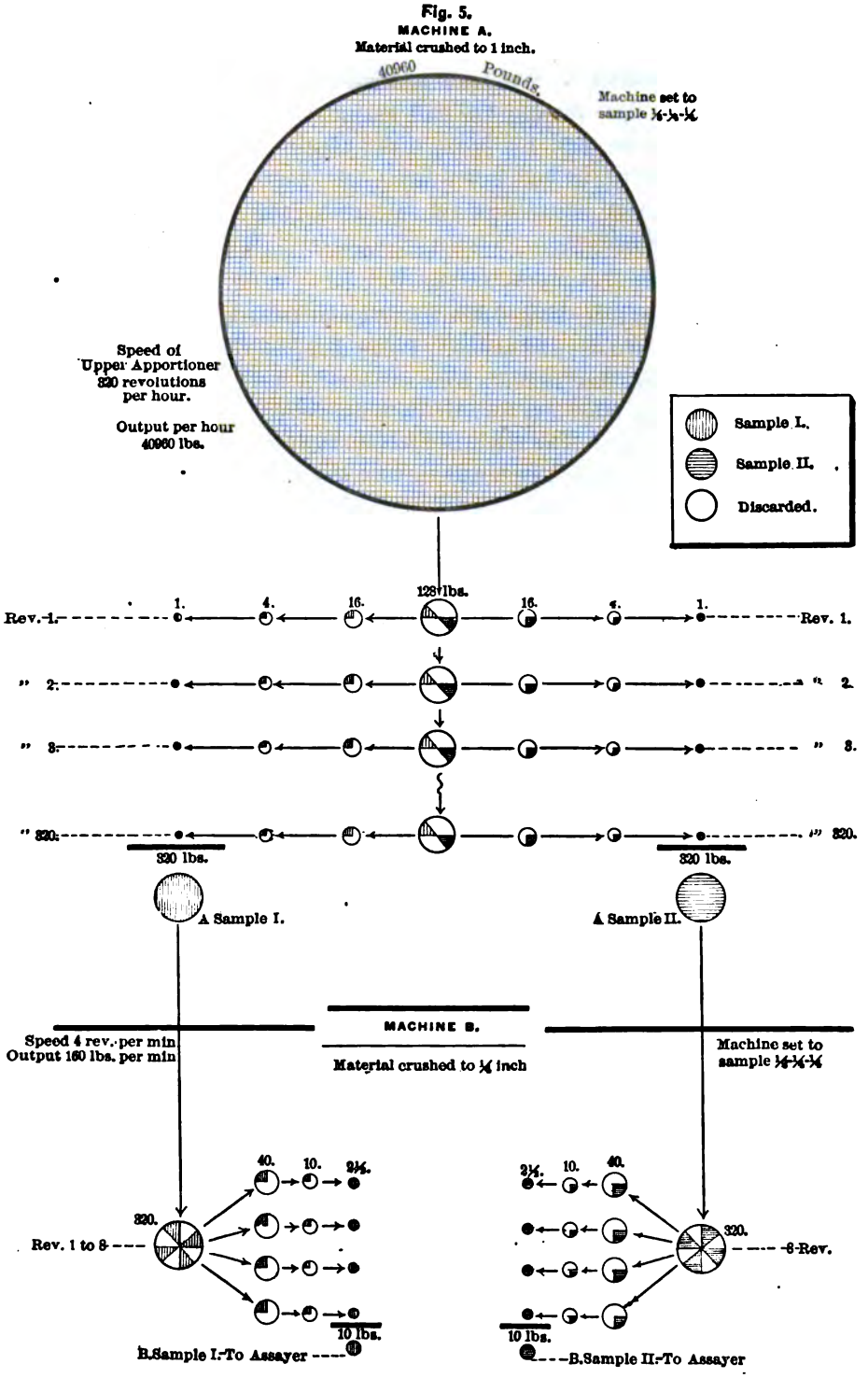
In the same way a 1-pound portion, as the third cut of the second sample, will find its way to the bucket Z_2.

This series of operations will occur at each revolution of the upper apportioner; and at the end of the hour each of the buckets Z_1 and Z_2 will contain 320 portions of 1 pound each, or a total final sample of 320 pounds, these two total samples being as independent of each other as though made at different times and places. It will of course rarely happen that this theoretical exactness of weights will obtain, which point will be considered later. Should the ore be of higher grade or more irregular in character, two or three or four of the spouts, M, may be set for each sample, giving final samples of 640, 960 and 1280 pounds respectively. The last case, in which nothing is discarded by the upper apportioner, is illustrated on the upper part of Fig. 6.

It will be noticed that only the discarded part of the sample is touched by the machine, the retained portions dropping nearly vertically and practically freely through the machine, until, *in a finished condition*, they reach the stationary receptacles R_1 and R_2 or the sample-buckets Z_1 and Z_2. The machine can have had, therefore, no influence on the constitution of the samples, and "coarse" and "fine" must be contained therein in the same proportion as delivered by the feed-spout, F.

It may be remarked in passing that the finer the material, the slower the feed and the greater the speed of the machine, the greater will be the distribution and presumably the better the samples. The conditions above given, however, are easily attained, depending only on the crushing-capacity at disposal, and have been found by experience to give satisfactory results, it being particularly desirable not to use a much higher speed. For light or wet ores it may be

Fig. 5.



MACHINE-SAMPLING.

necessary, in order to avoid an accumulation of material in the machine, to reduce the speed to half that given. This lower speed may of course be used for heavy materials also, the only practical difference between the higher and lower speeds (aside from the influence of centrifugal force) being the difference in the number of cuts made by the machine.

Machine B.

This machine occupies a floor-space of about 18 inches square, with a total height of about 36 inches. Being designed for cutting down the fine-crushed final samples from machine A, it gives but a single sample; but is otherwise similar in construction and operation to the large machine. The machine itself is shown in Fig. 4, its operation being graphically illustrated on the lower portions of Figs. 5 and 6.

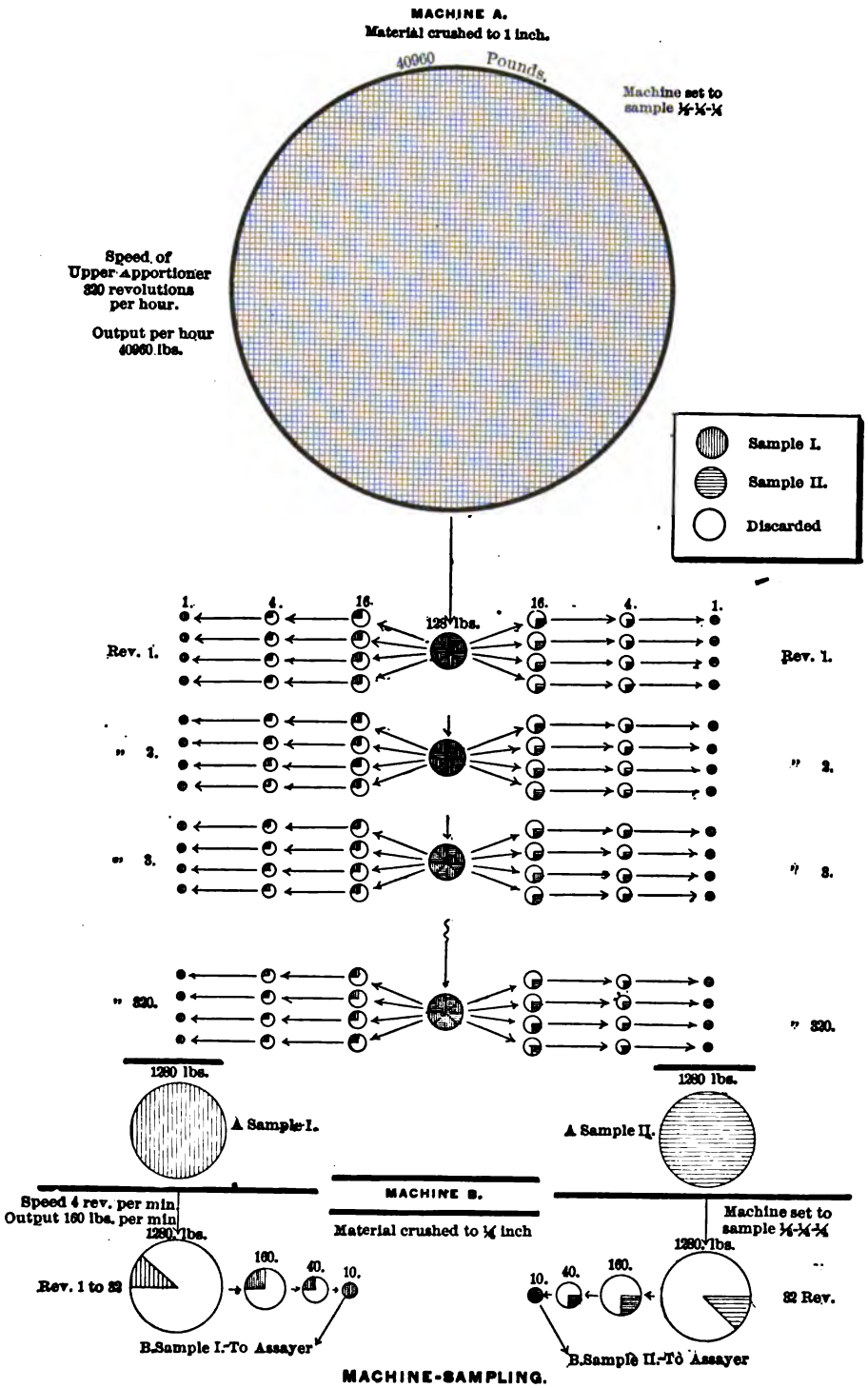
For the sake of comparison, a graphic illustration of hand-sampling is given in Fig. 7, which, like Figs. 5 and 6, shows an original sample of 40,960 pounds, cut down to two (duplicate) samples of 320 pounds each; these, after fine-crushing, being further reduced to final samples of 10 pounds each. These diagrams fully explain themselves, and no remarks are necessary, except to call attention to the very much greater number of cuts given by the machine than by hand, amounting for Fig. 5 to $320 \times 3 + 4 \times 3 = 972$ cuts, and for Fig. 6 to $320 \times 4 \times 3 + 3 = 3843$ cuts; while Fig. 7 shows only $7 + 5 = 12$ cuts.

Figs. 8 to 13, which further illustrate hand-sampling, fairly represent good practice. Fig 8 shows the first finished pile, containing the entire original sample, and also illustrates the method of delivering each shovelful of ore in such a way as to distribute it evenly over the entire surface of the cone. To do this successfully requires some skill; the material must not be thrown as in ordinary work, but must be tossed vertically into the air, so that it will fall squarely upon the apex and spread uniformly over the pile. Figs. 9 and 12 show the pile partially spread out, and the manner in which this is accomplished; Fig. 11, the pile completely spread out; and Fig. 10 the same pile, with the two discarded quarters removed. Figs. 12 and 13 together, indicate the manner of beginning a new pile.

Machine C, the Laboratory Sampler.

This machine occupies a space some 14 inches square, and about 14 inches in height. It was designed for cutting down the ground

Fig. 6.



MACHINE-SAMPLING.

final samples of the B machine, and for the sampling of the many small lots of material weighing from 20 to 500 pounds that are constantly being received by most assay-offices. Its operation is as follows (reference being made to Fig. 15):

The material is fed either by hand, or (with large lots) from a suitably supported bucket, into the funnel, F, the divider, D, being first set in rotation by hand, clock work or any convenient power. The divider gives, as will be seen by inspection of the drawing, eight cuts to the revolution, four being delivered to the funnel 1, and four to the receptacle 2; that is, with uniform flow and speed, it cuts the material in half. The divider may easily run 100 revolutions per minute, giving in that time 800 cuts, a very much greater distribution and division than can be secured in any other way. The rejected sample passes down the outlet, O_2, the retained portion through the outlet, O_1, both into suitable vessels. The retained portion, should it be too large, may be cut again and again, until of suitable size. The operation is very accurate and very rapid, being about as fast as the material will flow through a one-inch spout, and very much faster than is possible with hand-work or with sampling-tin, and of course not nearly as tiresome or expensive.

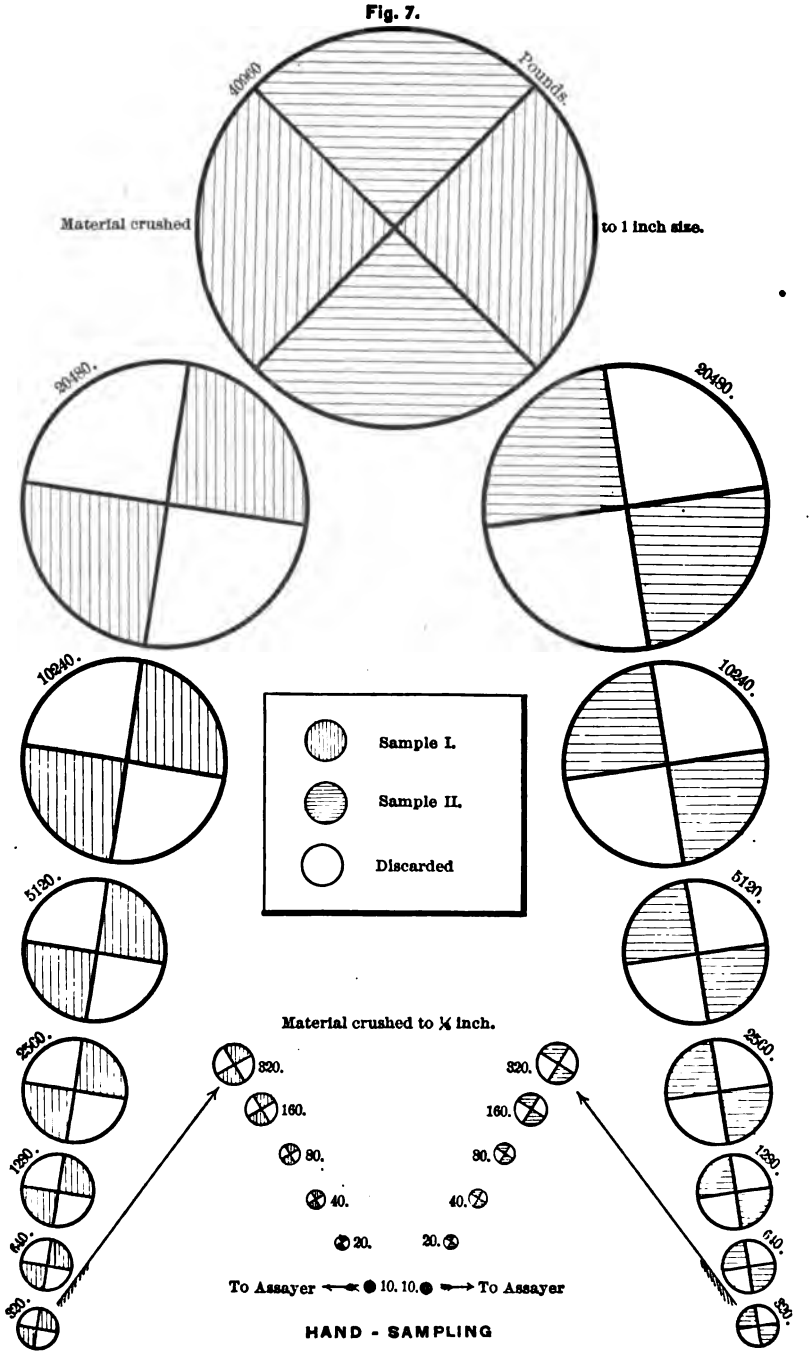
Mixer and Divider.

This apparatus, illustrated in Fig. 16, entirely obviates the tedious and often inaccurate methods (usually with oil-cloth and spatula) now in use, for mixing and dividing the ground samples of ore, matte, slag and similar materials. An experience of several months has shown a decided improvement in accuracy, speed, and general convenience over the old way, the saving in *time* amounting to fully two-thirds. The operation is as follows:

The ground material is introduced into the large covered funnel (mixer), the outlet being first closed by thumb or finger as may be most convenient. Funnel and contents are well shaken for a few moments, and then, with opened outlet, passed to and fro over the set of distributing funnels (divider) and bottles as shown. With very finely ground or very light material the flow may be assisted by a slight shaking or tapping with the hand. The little skill necessary is readily acquired.

To test the efficiency of the mixer, a lot of 6 assay-tons of litharge, 3 assay-tons of soda and three-quarters assay-ton of argol (9¾ assay-tons altogether) was taken, well shaken, and divided by weight into three

Fig. 7.


40960.
Pounds.
Material crushed
to 1 inch size.
20480.
20480.
10240.
10240.
Sample I.
Sample II.
Discarded
5120.
5120.
Material crushed to ¼ inch.
2560.
2560.
320.
320.
160.
160.
1280.
1280.
80.
80.
640.
640.
40.
40.
320.
320.
20.
20.
To Assayer
10. 10.
To Assayer
HAND - SAMPLING

lots of 3½ assay-tons each, and these charges were fused separately in crucibles. The resulting lead buttons weighed 53.436, 53.416, 53.398 grammes, which result should be conclusive as to the thoroughness of the mixing.

Tests in Practice.

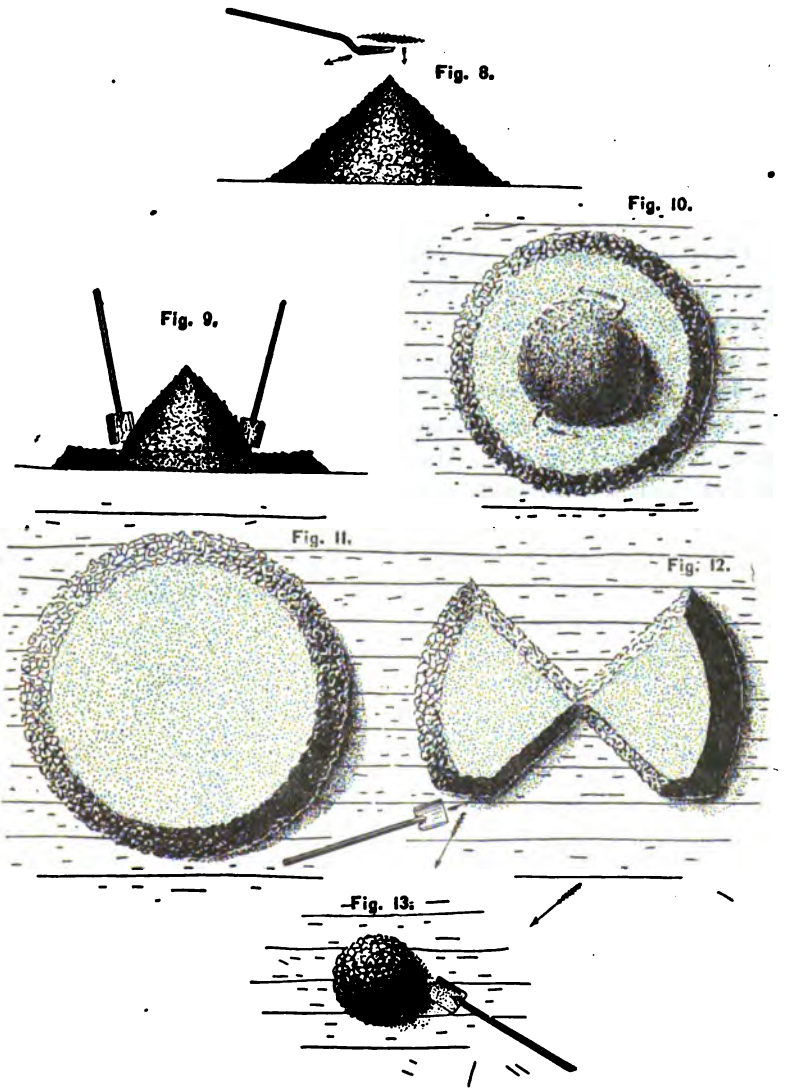
Having considered these various devices as parts of a general system of ore-sampling, it may be well to return to the investigation of the large machine, paying particular attention to the results obtained. That it is theoretically possible to do good work with this machine may be taken for granted; but actual use must demonstrate that it will operate satisfactorily under any ordinary working conditions. Let the records show what it has done.

The first machine built, and so far the only one, has now been in operation, at the works of which the writer has charge, for something over a year. It is set below, and takes its feed directly from, a No. 2 Gates crusher, whose capacity, large as it is, is very much less than that of the sampler. The product of this crusher will not exceed three-fourths inch ring-gauge in size. No attempt whatever is made to secure any special regularity of flow, the man at the crusher feeding as may be most convenient for himself. The discarded portion of the sample is removed by bucket-elevator to storage-bins, the two final samples (of 200 to 500 pounds each) alone remaining for further handling. These final samples are often obtained within half an hour after the last of the material is out of the car. They should be crushed to one-quarter inch or one-eighth inch ring-gauge and run through the smaller sampler. Having, however, heretofore had neither fine crusher nor small sampler, we have merely quartered them down by hand to, say, 20 pounds, passed them through a laboratory crusher, cut them down again with sampling-tin and "bucked" them; the ground material being finally mixed and distributed by means of the mixer and divider previously described.

Prior to the introduction of this machine, our works had treated 54 car-load lots (about 30,000 pounds each) of copper matte, on which double samples were made by hand. The average assay-contents of these 54 lots were: 7.88 ounces gold; 168 71 ounces silver; 55.24 per cent. copper. The average differences between the two samples of each lot were 0.43 ounce gold; 3.77 ounces silver; 0.71 per cent. copper.

Since the introduction of the machine, 22 lots of ore and 138 lots

of matte have been run, the latter being of the same general character as the hand-sampled matte, except that it did not, as a rule, carry so much "metallics." By reason of these "metallics" much



Hand-Sampling.

of this matte was very difficult to sample accurately, as will be easily understood. The weights of these 160 lots varied from 65 pounds to 42,000 pounds, averaging not less than 30,000 pounds.

Their average assay-contents were 0.71 ounce gold; 112.04 ounces silver; 51.75 per cent. copper.

The average differences between the two samples of each lot were 0.02 ounce gold; 1.19 ounces silver; 0.23 per cent. copper.

Reduced to percentages for the sake of comparison we find the average differences as follows:

	Gold. Ounces.	Silver. Ounces.	Copper. Per cent.
54 hand-samples, ...	5.46	2.24	1.29
160 machine-samples, ...	2.82	1.06	0.45

Certainly a creditable showing for the machine. The high percentage differences in the gold are accounted for by the multiplication of the inevitable error in weighing such small quantities.

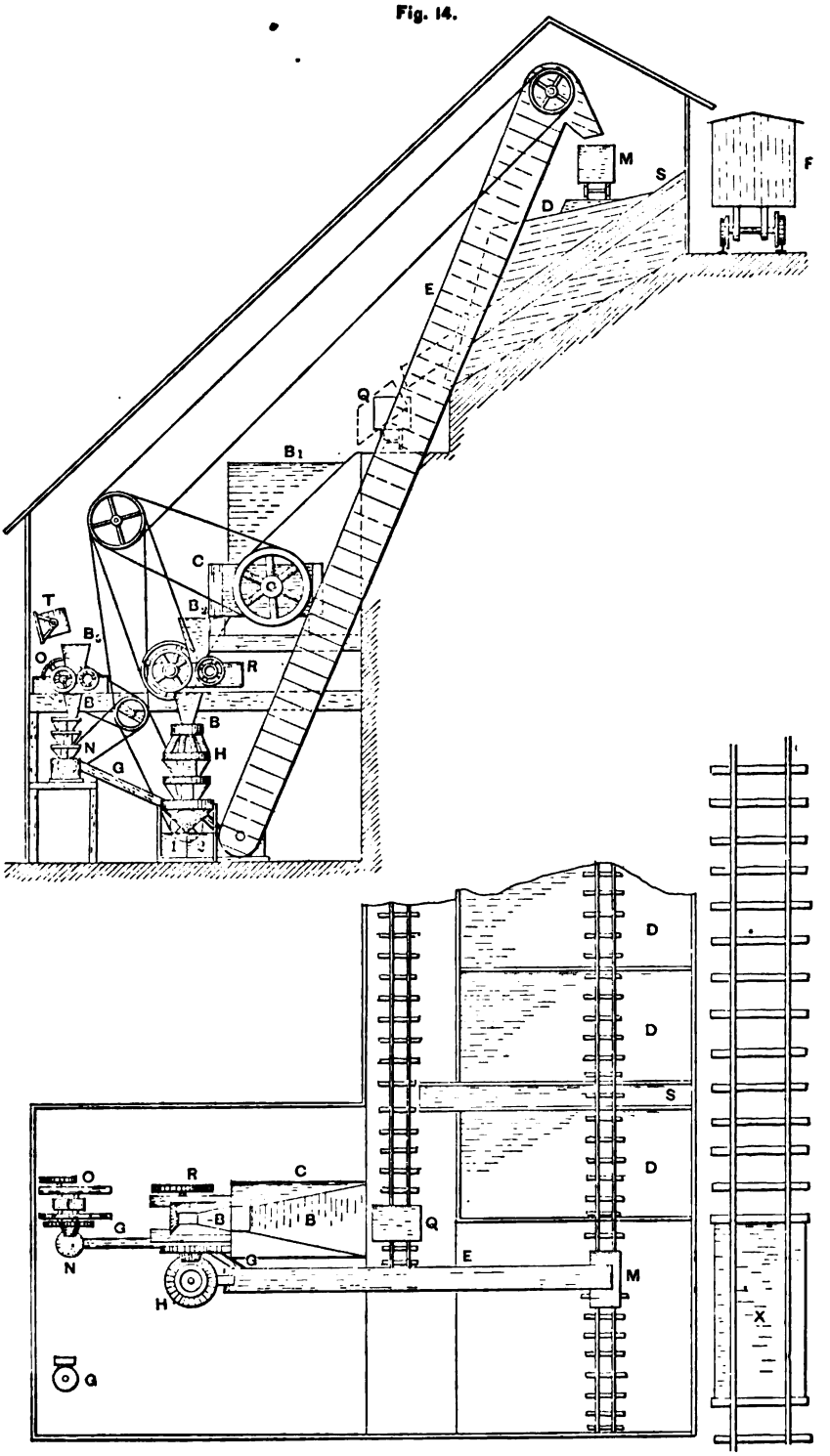
In all of this work the upper apportioner was set to take one-eighth of the entire material for each sample, or one-eighth by one-fourth by one-fourth, equal to one one-hundred-and-twenty-eighth, for each of the final samples. Had the machine operated as intended, each of the latter should have weighed, for a 40,000-pound lot, 312½ pounds. Such exact results, however, are very rarely attained, as will be seen from the following weights of the two samples from 40,000-pound lots. This list was taken at random, and is believed to be fairly representative:

Outer sample.	Inner sample.
170	413
526	194
473	191
190	327
300	315
471	211
238	360
462	447
101	166
500	529

It will be noticed that sometimes one sample is the larger and sometimes the other.

Within about these limits, there seems to be no regularity in the weight of the samples; generally speaking, however, the two samples, be they heavy or light, do not vary greatly from each other. Some of the samples show as great a difference in weight as 250 pounds, which, compared with the weight of the samples themselves, is very large; compared, however, with the weight of the original

Fig. 14.



Sampling-Works.

sample, say 40,000 pounds, it may be practically neglected. As frequently demonstrated by observation, it is certain that each time the sample shoots of the three apportioners come into proper position with relation to each other (that is, each time they line up as explained in the description of the machine), a portion of the material under treatment drops into one or the other of the sample-buckets, as the case may be. This portion will seldom be exactly the calculated quantity, but it will nevertheless be a practical sample of a small part of the entire lot, and consequently the total of these small samples will give a practical sample of the entire lot itself, whether the weights are theoretically correct or not.

When, therefore, there are two absolutely independent samples of this character, it is believed that an insight into the composition of the material is obtained that is possible in no other way. It may be remarked, in passing, that with hand-sampling the same number of cuts on each sample will give final samples with as great average differences in weight as are given by the machine. One cause for this irregularity in weight of the machine-samples is doubtless the lack of uniformity in the feed, this being particularly the case when the samples vary between themselves and when they weigh less than the theoretical quantity. The principal cause is less apparent, and is believed to be due to the manner of flow of the crushed material, as is explained by the following considerations.

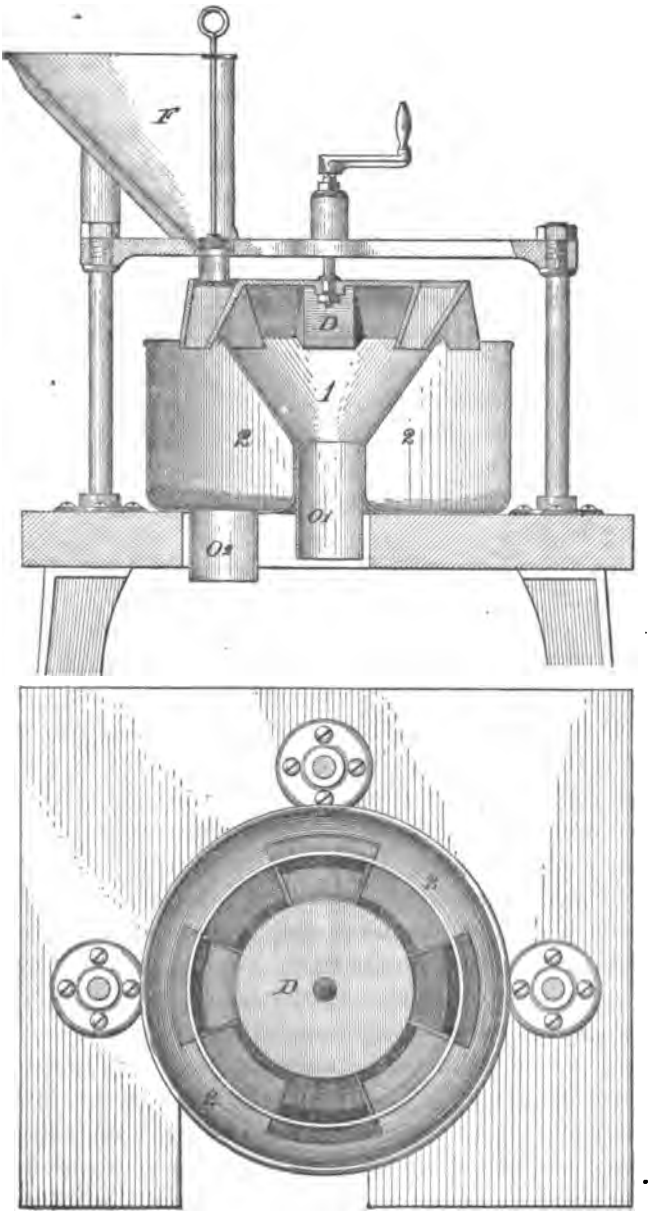
In passing through the machine, the material may be conceived:

1. As a shower of particles, nearly uniform in size, and completely filling the space. In this case the upper apportioner would take one-eighth, the intermediate apportioner one-fourth of the one-eighth, and the lower apportioner one-fourth of the one-fourth. That is, the machine would operate as intended, and each final sample from a 40,000-pound lot would weigh 312½ pounds.

2. As concentrated into a single vertical line for each sample. In this case the speed-relations of the three apportioners with reference to each other would cease to be of influence. For instance, the upper apportioner makes 5 revolutions in one direction, while the intermediate apportioner makes 15 revolutions in the other; that is to say, moves relatively four times as fast, and, under the first case, one-fourth of the "shower of particles" which passes through the sample-shoot of the upper apportioner would be cut out by the sample-shoot of the intermediate apportioner. Under the second case, the "line of concentration" would be cut one-eighth by the upper apportioner, and this one-eighth would be cut one-third by

the intermediate apportioner, exactly as though the upper appor-

Fig. 15.



Laboratory-Sampler. Size C.

tioner revolved in the same direction as the intermediate, and one-

third as fast (that is, as 5 to 15). Under the first case, the motion of the upper apportioner influences the manner in which the stream is cut by the intermediate apportioner. Under the second case, it does not; and so far as the cutting of that portion of the stream which reaches the intermediate apportioner is concerned, the upper apportioner might as well not exist. The same reasoning would apply to the lower apportioner, and we would have for the cuts under this second case one-eighth by one-third by one-third, equal to one-seventy-second; that is, each sample from a 40,000-pound lot would weigh 556 pounds.

In practice, the action of the material would lie somewhere between these two extremes, generally more nearly approaching the former. It is believed that a large share of the variation in weights is due to this cause.

It is obvious, however, that in either case a practically true sample would result, so that this whole question has little or no practical importance.

As showing the general adaptability of the machine, two extreme cases may be given:

I. As a test of capacity, 42,000 pounds of 50 per cent. copper matte, after preliminary crushing, was fed by hand directly to the machine, requiring the steady work of three good shovelers. The entire lot was put through in 66 minutes, the resulting samples weighing 501 and 300 pounds respectively.

II. 9900 pounds of oxidized copper-ore, carrying about 50 per cent. of copper and 7.5 per cent. of moisture (wet enough to ball in the hand), was fed directly from the crusher in 8 hours (as fast as the crusher could work, on account of the character of the ore), the resulting samples weighing 117 and 110 pounds.

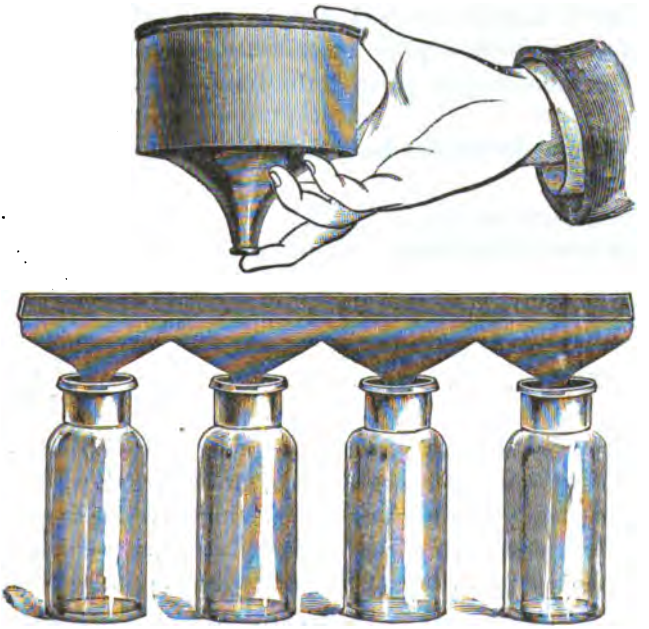
In both cases the assay-results were satisfactory. From the above statements it will be recognized that in the somewhat varied and often difficult conditions under which this machine has heretofore worked, it has at least equalled in accuracy the best of hand-sampling, which is the accepted standard everywhere.

What it will do with the innumerable kinds and grades of ores and other products which form by far the larger portion of material subjected to sampling, is of course not definitely known at present. It is fair, however, to judge of the possibilities of the future by the performance of the past, and to state the general belief that, with any material that can be crushed (being neither too wet nor too metallic), or that is fine enough to need no crushing, the machine

will give results equal in accuracy to any hand sampling at a small fraction of the expense, and in very much less time.

Nor must the fact be overlooked that the double samples, furnished without extra expense by the machine, offer an excellent safeguard against "salting" and against errors of every kind. The practical impossibility of "salting" the samples from this machine is shown by the fact that both samples would have to be salted alike, and as the weights of the two are never exactly the same, any attempt of the kind would be almost certain to result in detection. Moreover,

FIG. 16.



Mixer and Divider.

the result is obtained so quickly that it is easy to control any possible difference, either of fraud or of error, by re-sampling the lot while it is still intact.

It may be objected to the double samples that they increase the assay-work. This may be true, and if true should not be deemed a drawback, since the accuracy of results is increased in the same proportion. But it will not necessarily increase the work, for a single assay on each one of the two samples is certainly far better than a duplicate assay on only one sample. With proper appliances, as in-

dicated in this paper, the extra labor in preparing the samples is very small.

In computing the relative cost of sampling with this machine and by hand, it is best to begin with the ore as it issues from the crusher, and to end with the delivery of the crushed and sampled material into its appropriate bin.

In the latter case, this delivery must be accomplished almost entirely by hand-labor (generally with wheel-barrows), while with machine-sampling it is done almost entirely by elevators or conveyors.

The large machine will readily handle 20 tons of material an hour, and it may be assumed that, for each lot of this size, the man in charge will require another hour to clean the machine and to run the samples through the small machine to their final size.

The expense for the two hours would be:

Two hours for one man, say	$0.50
Power and lubrication, say	0.05
Wear and tear, say	0.05
Total for 20 tons,	$0.60
For 1 ton,	$0.03

This has been shown by experience to be a very liberal estimate. Power and lubrication cost much less than is charged, while the wear resulting from a year's work of the existing machine is practically nothing, although in this first machine all the parts are much lighter than in the ones now building. Moreover, the entire time of the man has been charged to sampling, whereas it is perfectly feasible to have him feed the crusher or do other work; in short, one man is sufficient to do all of the actual sampling-work of a mill handling not over 100 tons of sample or, say, 500 to 1000 tons of ore a day.

For the hand-sampling of 20 tons (of original sample) there would be required at least:

Two shovelers for ten hours at 20 cents per hour,	$4.00
One wheel-barrow-man for ten hours at 20 cents per hour,	2.00
Total for 20 tons,	$6.00
For 1 ton,	$0.30

This expense should give double samples. For a single sample the cost would be:

One shoveler for twelve hours at 20 cents, ...	$2.40
One wheel-barrow-man for ten hours at 20 cents, . . .	2.00
Total for 20 tons, ...	$4.40
For 1 ton, ...	$0.22

That is, the machine delivers double samples at one-seventh the cost of single samples by hand, or at one-tenth the cost of double samples by hand. Nor does this saving, large as it is, by any means represent the total gain from the use of this machine, for the numerous advantages elsewhere enumerated undoubtedly lead to important reductions of expense that are not so readily determinable. To all of which is added the satisfaction of always knowing exactly what is being done, and the additional satisfaction of knowing it promptly.

For the hand-sampling of 100 tons of original sample a day, a floor-space of at least 2000 square feet would be required, while for machine-sampling not over 200 square feet is necessary. An inspection of Fig. 14 will give an idea of the possibilities in this direction, a properly constructed works being there shown in elevation and ground-plan. An equally efficient plan may be made for level ground by arranging to deliver the entire sample from either the large car, F, or the small car, Q, directly to the crusher, which, with its rolls, should be set above the sampler if practicable. This arrangement avoids the necessity for bucket-elevators between the crushing-machinery and the sampler, thereby doing away with a great deal of vexatious cleaning and of risk.

Where it is not possible to escape this use of elevators, the writer has designed a special construction in which both cleaning and risk are reduced to a minimum. The railroad car, F (Fig. 14), after passing the scales, X, is unloaded into any one of the bins, D, of which there may be any number. As the unloading proceeds, the portion required for sample (be it one-tenth or one-fifth or the entire lot) is delivered through troughs, S, which are located between each two bins, D, into one or more cars, Q, which, in turn, deliver into the hopper, B_1. From B_1 the material passes the crusher, C, the rolls, R, and the large sampler, H, the discarded portion (all but a few hundred pounds) passing into the elevator, E, and thence by cars, M, to the same bin as the rest of the lot, or to any other de-

sired point. The samples in the buckets 1 and 2, weighing but a few hundred pounds each, are elevated by chain-tackle to the tilting-frame, T, whence they pass through the hopper, B_3, the fine rolls, O, and the small sampler, N, the discarded portion going to the elevator, E, and the final samples, of a few pounds each, to the sample grinder, G, and thence to the assay-office.

Such a plant will secure not only the greatest economy, but also the greatest security from errors and from malicious interference with the samples.

As showing the adaptability of these machines, it may be mentioned that one of the largest smelting-works on this continent is now putting in a set of three, two large and one small. The entire sampling-plant is to be located adjacent to the main engine, the machines themselves being in a room excavated below the level of the crusher-floor. An ordinary sampling-floor could not have been so located by reason of the room required, and would have required, moreover, an independent engine. This saving in building and machinery will probably more than equal the entire cost of the sampling machinery. Either one of the large machines would be abundantly sufficient for all the work; but two are to be employed, one for oxide and the other for sulphide ores, it being desirable to keep these two classes of material entirely separate, and cheaper to employ two machines than to do the transportation and handling which would otherwise be necessary.

In conclusion, it may be emphasized that this machine is no longer an experiment. Its record of over one million dollars' worth of material purchased entirely on its samples, and from men who would tolerate no loose work, places it fairly in the category of accomplished facts.

THE FLORENCE OIL-FIELD, COLORADO.

BY GEO. H. ELDRIDGE, U. S. GEOLOGICAL SURVEY, WASHINGTON, D. C.

(Glen Summit Meeting, October, 1891.)

INTRODUCTION.

THIS sketch of the Florence oil-field, presented to the Institute by permission of the Director of the U. S. Geological Survey, is accompanied with a map, Fig. 1, the topography of which has been taken from a field-manuscript of the Survey. The scale of the map and

the sections, Fig. 2, as herewith printed, is 3 miles to the inch. The symbols employed to indicate geological formations, etc., are:

a, Archæan; S, Silurian; C, Carboniferous; T, Trias; J, Jura; Kd, Dakota (Cretaceous); Kb, Benton; Kn, Niobrara; M, Montana (including Pierre and Fox Hills); Kl, Laramie; q, Quaternary; O, Oil-zone; W, Wells.

The dots in the vicinity of the town of Florence indicate the positions of the wells.

The outlines of the Tertiary formations have been omitted intentionally. The areas of these formations are very small.

Geographical Position of the Field.—The Florence oil-field lies in Fremont Co., Colorado, along the valley of the Arkansas river in close proximity to the eastern base of the Rocky Mountains. Its present center of production is near the town of Florence, 92 miles a little west of south from Denver, 30 miles in a west-northwest direction from Pueblo, and 8 miles below Cañon City—the point of exit of the river from the range. Its rail communication east and west is by the Denver and Rio Grande, and Atchison, Topeka and Santa Fé systems.

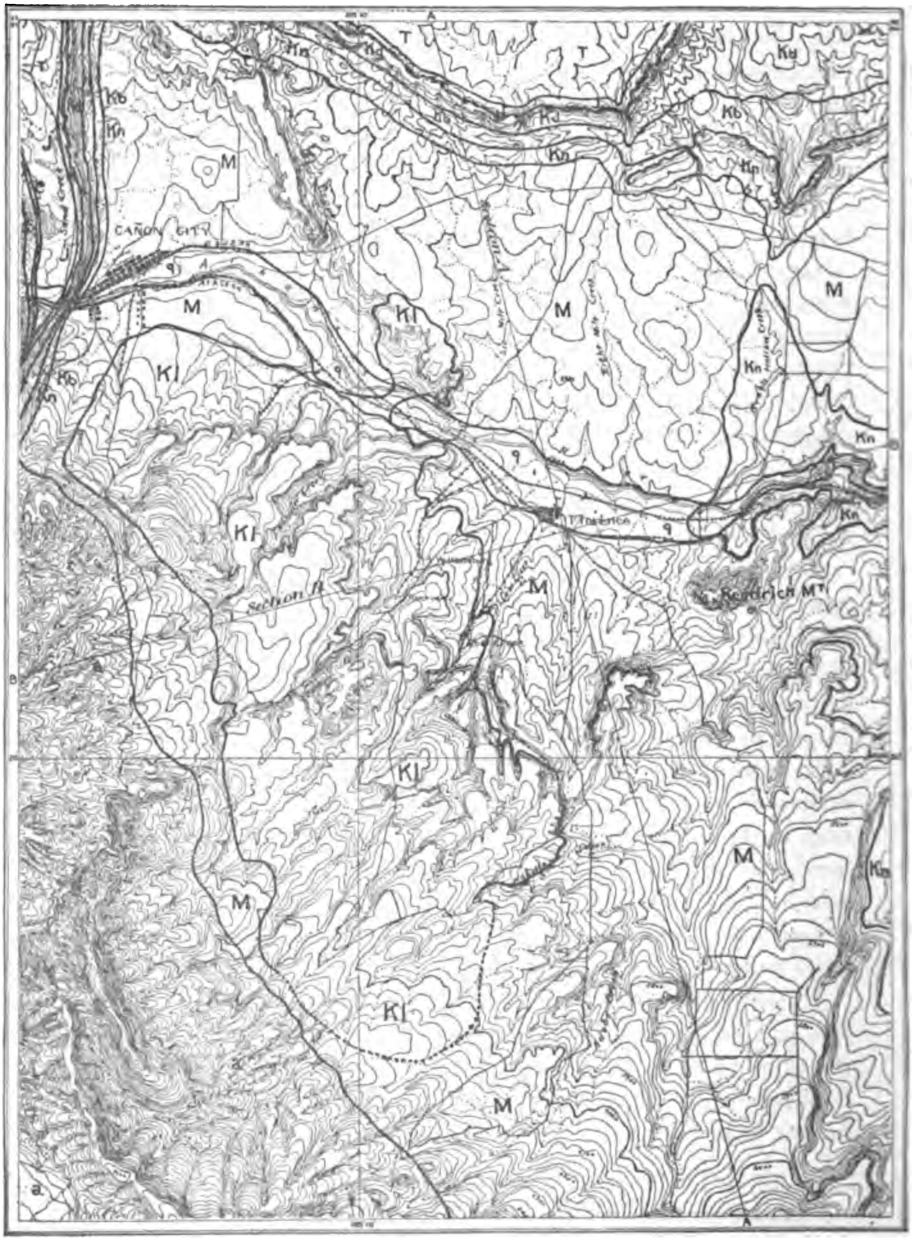
*Development of the Petroleum Industry.**—Indications of petroleum were first observed in the course of early explorations by an army officer in the cañon of Oil creek, at a point about six miles northeast of the present site of Cañon City. The occurrence was in the form of seepage in the creek-bottom, at the foot of bluffs of Jurassic age. It was in this locality, in March, 1862, that the first active prospecting was begun by a Mr. Cassidy, who sank six wells to a depth of from 60 to 90 feet and two to a depth of 400 feet, but found oil only in the strata near the surface. This was pumped, crudely refined, and shipped to Denver and other territorial markets, where it sold for $1.25 to $2.85 per gallon, the price at one time reaching $5 per gallon. The sales up to 1870 are said to have amounted to 3000 gallons.

In 1866 and 1873 two other wells were sunk to depths of 312 feet and 342 feet respectively, without developing any other than the supply already encountered a short distance beneath the surface.

Subsequently to this—1877 to 1885—three companies, the Arkansas Valley Oil and Land Company, commonly known as the Cas-

* For much of the earlier history of development, acknowledgment is due Prof. M. C. Ihlseng's *Report on Oil-Fields of Fremont County*, in the Annual Report of the Colorado State School of Mines for 1885.

Fig. 1.



Map of the Florence Oil-Field, Colorado.

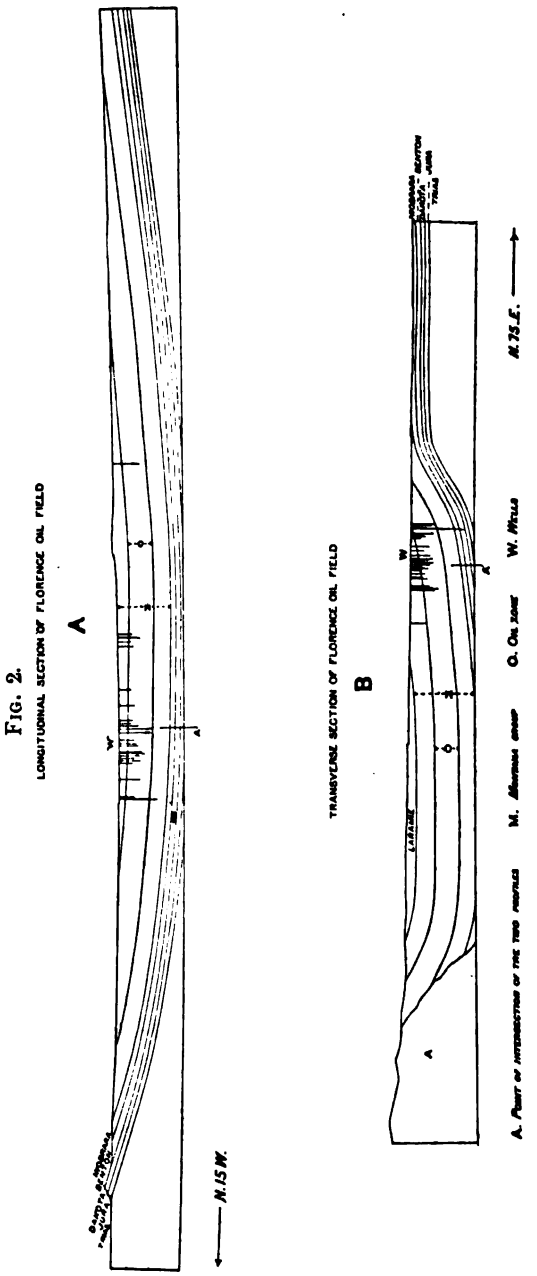


FIG. 2.

Sections on Lines A A and B B of Fig. 1.

sidy Company, the Land Investment and Coal and Iron Company, afterwards becoming through one or two transitions the Colorado Oil Company, and the Cañon City Oil Company, undertook explorations on the prairie bordering the Arkansas river below Cañon City. These were the first explorations in strata of the Montana group of the Cretaceous, the group that alone affords the yield of to-day.

The Arkansas Valley Company was developed from individual interests which embraced two wells: one experimental, located a half mile below Cañon City, to the south of the river, bored in 1877 and abandoned at a depth of 900 feet without discovering oil; the other near the present town of Coal Creek, sunk in 1880 for water, in which oil was encountered at a depth of 1200 feet, but which was abandoned on account of loss of tools and litigation. The company wells were located about 1¼ miles south-southeast of the present site of Florence, were 6 (7?) in number, ranged in depth to 1740 feet (one to 3012 feet), one of which was dry, the others producing from ½ barrel to 20 barrels (one, according to account, 100 barrels) per day, some of which are yielding at the present time.

The Land Investment and Coal and Iron Company, incorporated in 1882 and subsequently dissolved, was taken up by D. G. Peabody, its former president, and finally became the Colorado Oil Company of 1885–6. The wells of this company in the latter year numbered twelve, were with one exception within ¾ mile of the site of Florence, ranged in depth to 1700 feet, and included several good producers, although the majority were small in yield. This company and the Arkansas Valley Company were in the year 1887 absorbed by the United Oil Company of the present day.

The Cañon City Oil Company was incorporated in May, 1883. Its single well is located in the vicinity of Oil creek about 3 miles east-northeast of Cañon City, and 6½ miles northwest of Florence. Oil is reported as having been encountered at a depth of 300 feet, followed at a short interval by salt water which eventually caused the abandonment of the hole; a depth of 1700 feet, however, is said to have first been reached.

The companies at present in the field are, in order of magnitude: the United, the Florence, the Rocky Mountain, and the Triumph, the first two having been established about the year 1887, the others more recently, the Rocky Mountain Company being the youngest (1890).

In addition to the wells of these companies, there are distributed

over the field at considerable distances from the present center of production, a number of holes that have been drilled with varying success by individual enterprise.

Topography.

The Florence oil-field is confined within the limits of a synclinal basin. It is elliptical in area, the major axis having a trend of N. 15° W. and a length of about 14 miles, the minor, a length of 8 miles. Its topographical features are about equally attributable to structure, erosion, and rock-textures. The eastern rim of the basin, never so completely developed as the other portions of the periphery, has been considerably abraded by erosion. This is especially the case north of the Arkansas river. To the south of the river, however, the rim is still clearly visible in the gentle rise of land to the southeast, its exterior edge being especially accented by a sharp drop in contours at the locally eastern limit of the Montana group. Southward this rim passes directly into a minor though well developed ridge that juts at an acute angle from the main range of mountains forming the western boundary of the synclinal. Northward the eastern rim passes into what might at first be regarded as the western spur of a double échelon-fold, the lofty mass of mountains involved in the flexures carrying the face of the range fully twenty-five miles to the east, and presenting the finest illustration of the échelon type of topography and its manner of development, to be found in the entire length of the Colorado range.

In the western half of the field and south of the Arkansas river lies a conspicuous mesa, co-extensive with the Laramie formation, which is here composed of a succession of heavy bedded sandstones that on the northern, eastern, and southern periphery of the mesa form abrupt bluffs overlooking the rolling prairies of the Montana. There is also an outlier of the Laramie to the north of the river, two and a half miles northwest of Florence, known as "Castle Rock."

Erosion together with preservation beneath caps of Quaternary gravel has given existence to several other prominent heights throughout the field but in none of them are found strata of younger age than the Montana.

The drainage of the area is by the Arkansas river and its lateral tributaries, the latter carrying but slight surface flows.

GEOLOGY.

Description of the Formations.

The stratigraphy of the Florence oil-field and its confines embraces formations of Archæan, Silurian, Carboniferous, Triassic, Jurassic, Cretaceous, and Tertiary ages. Of these the Cretaceous formations are of chief concern in the present study; they comprise the Dakota, the Benton and Niobrara (together constituting the Colorado group), Pierre and Fox Hills (constituting the Montana group), and the Laramie.

On the northern and northern third of the western edge of the field the sedimentary formations from the Colorado down lie in a fringe along the Archæan and dip quite uniformly 30° beneath the prairie; a short distance south of the Arkansas river, however, structural peculiarities assert themselves, and from this point southward to beyond the limits of the field the Archæan is in direct contact with the Pierre shales of the Cretaceous. The formations younger than the Colorado lie wholly within the prairie region.

The Older Formations; the Laramie and the Tertiary.—The Archæan consists of a series of granites and schists; the Silurian and Carboniferous, of limestones; the Trias, of the well-known "Red Beds" of the Rocky Mountain region, a series of massive beds of coarse conglomerates and sandstones in its lower division, of finer brick-red sandstones, shales, and interspersed limestones in its upper; and the Jura, of shales, limestones, and occasional sandstones. Passing for the present directly to the Laramie, this formation comprises a series of heavy bedded sandstones with intercalated shales, and in the lower portion workable beds of excellent coal. In many parts of Colorado the proportion of shales to sandstones is much greater than here. The Tertiary formations are two; the lower, a series of coarse conglomerates and sandstones derived from the older formations, 250 feet to 300 feet in thickness; the upper, a succession of conglomerates, sandstones, and shales of eruptive *débris*. The former probably corresponds to the Arapahoe formation of the Denver field, the latter to the Denver formation of the same region. The areas occupied by the two Tertiaries are small and isolated.

The Cretaceous formations have the same general characteristics in the present field as elsewhere in the Rocky Mountain and prairie regions.

The Dakota Formation.—This, the basal member of the series, is essentially a sandstone formation between 200 and 300 feet thick, divided into two or more benches by bands of fire clay or other

argillaceous matter, the lowest bench having a prominent and persistent zone of characteristic conglomerate at its base. The material of the sandstones is a white quartz, of fine to medium grain; it is in many localites infiltrated with a siliceous cement which imparts to the rock a quartzitic appearance. The usual color of the Dakota sandstone is white or grayish-white, but it may be darkened several shades by the presence of iron or disseminated asphalt. This latter feature is not of infrequent occurrence in the West, and in several localities the bituminous matter seeps from the rock in well developed springs. In the Florence field the formation is heavily charged with mineral water which rises to the surface in copious flows wherever the strata are tapped by drilling.

The fossils of the Dakota in Colorado are chiefly confined to plant remains.

The Benton Formation.—This is a formation of black argillaceous shales, passing by transitional beds into the Dakota below, and the Niobrara above, though into the former in a manner more or less abrupt. Its thickness, like that of the other marine Cretaceous formations in the West, varies considerably from point to point, but is generally between 400 feet and 800 feet, in the present field falling somewhat below the lower figure. In addition to its leading characteristics—its black or leaden-hued color, argillaceous composition, and shaly nature—it is marked by the occurrence, here and there, of a few narrow and intermittent bands of fossiliferous limestones, more or less bituminous; by the frequent presence at certain horizons of concretionary, clay ironstones; and by the occasional appearance of a few narrow bands of impure argillaceous sandstones. As the summit of the formation is approached, the clays become more calcareous, and the limestone more prominently developed, the latter often bearing a distinct resemblance to the lower layers of the important bed at the base of the Niobrara, though rarely attaining a thickness of over four or five feet. Among the characteristic fossils of this formation, *Ostrea congesta* and *Inoceramus problematicus*, together with Selachian teeth are especially well represented, the last particularly in the more bituminous layers.

The Niobrara Formation.—The thickness of this formation varies throughout the West from a mere trace to approximately 400 feet, approaching the latter figure in the present region. When fully developed it is separable into three divisions: the lowest, a bluish-gray, hard, coarsely laminated limestone of an average thickness of 40 feet; the middle, a succession of gray marly clays, 100 feet; the

uppermost, a series of calcareo-arenaceous clays of various shades of yellow or buff, remarkable for the presence of "alkali salts" and containing several bands of impure, yellow, saccharoidal, fossiliferous limestone. The basal member, from its general persistency and the prominence of its characteristics, forms an admirable datum-level for reference in the study of Cretaceous stratigraphical geology. The life of the Niobrara includes three especially prominent molluscan forms, *Ostrea congesta*, *Inoceramus problematicus*, and *Inoceramus deformis*, the last a very characteristic species; fish integuments are also common.

The Pierre Formation.—This formation is, in the main, a great body of leaden-gray clays, carrying from bottom to top lenticular bodies of impure limestone, and, in several localities, midway in the series of beds, a zone of yellowish, quartzose, and more or less calcareous sandstone of a thickness varying from 100 feet to 200 feet. In the Florence field no evidence of this sandy zone exists, but at several points in the vertical scale there can be noticed a tendency of the clays toward a fine, arenaceous composition.

Though the formation attains the exceptional thickness of over 7000 feet in the vicinity of Denver, Colorado, it generally falls far below this figure, rarely attaining 1500 feet, and frequently only 700 feet or 800 feet. In the Florence field the thickness is probably about 4450 feet. The clays are in general of remarkably uniform texture; often show a tendency to concretionary structure, though always clearly and evenly stratified; are plastic in a noticeable degree, and have a minor distribution of lime, gypsum and "alkali salts" throughout. The lenticular bodies of limestone occur promiscuously through the formation and carry the bulk of its fossils. The dimensions of these bodies vary between 2 and 6 feet in the direction parallel to the bedding planes of the clays and between 6 inches and 2 feet in the direction normal to these planes. Their composition lies between that of a clay with very little carbonate of lime, and a very pure limestone, generally inclining to the more calcareous variety; they are frequently reticulated with narrow calcite seams. In many of the limestone bodies, and occasionally in the clays, minute particles of carbonized vegetable tissues abound. The life of the formation is abundant, but its especially prominent forms are mainly included in the genera *Inoceramus*, *Cucullæa*, *Nautilus*, *Placenticeras*, *Baculites*, and *Scaphites*. As the summit of the Pierre is approached the clays very gradually become more and more arenaceous, though never losing their distinctly argilla-

ceous composition until the indefinite line between this and the overlying formation is passed, when the material becomes as marked an argillaceous sandy shale as it was before the reverse.

The Fox Hills Formation.—Excepting its persistent and distinctive cap of yellow fossiliferous sandstones of a thickness of from 10 to 50 feet, this formation embraces a series of fine arenaceous shales with a large percentage of clay, varying in thickness between 300 feet and 1000 feet, according to locality. In the Florence field their thickness has been estimated at 450 feet. Limestone concretions similar to those in the Pierre occur here and there throughout the shales. There is a transition in life as well as in sediment from the Pierre to the Fox Hills, and only at the summit of the latter formation does it become completely differentiated.

The Thickness of the Montana Group (Pierre and Fox Hills) in the Florence Oil-field.—This has been taken at 4900 feet, of which 450 feet have been allotted the Fox Hills, leaving 4450 feet as the thickness of the Pierre. The total thickness of the group, 4900 feet, is based upon the following evidence, an actual measurement on a definite line being found impracticable. In one of the wells in the eastern portion of the area at present developed, Pierre shales were alone passed to a depth of 3300 feet. The line of union of Pierre and Fox Hills indisputably lies to the west of the point at which the hole was drilled, and had the Niobrara been entered it would have immediately been recognized by its color and difference in chemical and physical composition from the Pierre. The base of the formation was arbitrarily established 200 feet beneath the bottom of this well. This additional distance in depth was taken: first, because no evidence existed of the base of the formation having been reached in the well, and it was therefore necessary that some increase should be made; secondly, 1¾ miles east of the well in question are two others, yielding artesian water, the start in drilling them having been made at a horizon not over 50 feet beneath the summit of the Niobrara; thirdly, no dips were anywhere observed of greater degree than sufficient to carry the strata into the positions assigned them upon the above basis. The observed surface dips were usually much less than required under the above conditions, but flexures in the strata unquestionably occur to compensate any lack in this direction. At several points in the field such flexures were apparent. The thickness of the remainder of the Montana from the above well westward to the clearly defined line of the base of the Laramie is

determined from careful observations of dip, distance, and difference in altitude.

It is rarely that the Montana group can be separated into its component formations at any distinct line, but there is almost always a change of aspect both in its composition and its fossils within a zone of 100 feet or 200 feet that enables one to decide as to the presence of the one or the other. Such is the case in the present field, and at several points on the higher lands, in the vicinity of a line passing N. 15 W. a half mile west of Florence, the rocks have a distinct Fox Hills aspect. On the visible dips (mostly 5°) between this line and that of the base of the Laramie to the west, with the requisite allowance for difference in altitude between the two, 450 feet has been regarded as the nearest possible approximation to correctness.

The thickness assigned the Pierre formation is one of difference between that determined for the Montana as a whole and that of the Fox Hills.

Mr. Johnson, Superintendent of the Florence Oil Company, has determined a thickness of 4500 feet for the Montana group, based upon measurements made by him in the foot-hills.

The Structure of the Field.

The Florence oil-field lies within a synclinal of elliptical area, the major axis of which has a trend of N. 15 W. and passes somewhat east of the center of the region occupied by the Laramie, the minor axis passing across the field about a mile south of the town of Coal Creek. The structure of the field and the related region that adjoins it on the east, has been brought about by the combined influence of what is in reality a comparatively slightly developed secondary mountain fold and a fold *en échelon*, the latter lying to the east of the former.

The secondary fold appears in the anticlinal arch crossing the Arkansas river at the mouth of Brush creek, its axis passing just east of Kendrick mountain. Its southern limit is sharply emphasized in a fold jutting to the northward, at an acute angle from the main front range, at a point about thirteen miles due south of Florence. Northward the arch passes into what might be at first regarded as the western spur of a double échelon fold, but which is, however, distinct in structure, though in a measure related to, and probably of synchronous development with the true échelon fold to the east.

The fold *en échelon* which lies wholly to the east of the area

mapped for this paper, but which may be referred to on sheet XIII. of Hayden's *Atlas of Colorado*, has had an extensive influence upon the superficial relations of the Cretaceous formations east of the secondary fold just described. Its axis lies well out on the prairie, and the effect of its arch, rapidly flattening as distance south is gained, is manifest in the outcrops of one or another of the lower three formations of the Cretaceous, not only along the river from the eastern rim of the oil-field to a point several miles below Pueblo, but also for a considerable distance to the south of the river. From within this great area of its influence the Montana group has for the most part been removed, and with it the oil-zone of the Florence field.

Questions as to the presence of faults and fissures within the limits of the Florence oil-field have repeatedly arisen. That slight local displacements of strata have taken place can hardly be doubted, especially in the shales of the Montana, which show numerous flexures, but the occurrence of faults of considerable displacement, affecting the series of formations underlying the field, is negatived by the enormous flows of artesian water from the Dakota sandstones wherever pierced. The points of entrance of such waters are at the outcrops of the formation at the base of the mountains, and their constant and copious discharge precludes all possibility of interruption. Again, such slight displacements in the body of shales as may have occurred, no matter how widely distributed, can hardly have left the series open-fissured in such a manner, for instance, as to form reservoirs for the reception of infiltrating oil, a view held by some, for the plasticity and yielding nature of the shales would, on account of accompanying pressure, very generally prevent the formation of such cavities. The possibility, however, of the occasional existence of open fissures within the Pierre is not denied, but they must be small and of slight extent both in vertical and horizontal range. Their existence cannot affect the general views expressed beyond in regard to the nature of the occurrence of the oil, for in any event the fissures would be but reservoirs of a more open condition than the arenaceous bodies already containing the oil—a large pore in the formation—into which the oil would filter as into the sands. Upon establishing communication with the surface, they would become merely a part of the bore which pierced them. Of the existence, however, of such fissures, and encountering them in drilling, there is no authentic evidence whatever.

The Horizon of the Oil.

Lithological Character.—The distribution of the materials of the Pierre and its included oil-bearing zone, both in lateral and vertical range, is an important consideration in reference to the source and storage of the contained oil. The formation is chiefly one of argillaceous shales, with occasional local developments of sandy strata of various degrees of coarseness. In the Florence field the coarser varieties of sand are wanting, but there often exists an extremely fine arenaceous matter intimately mixed with the clay. The deposition of these arenaceous materials has been either as a succession of beds passing laterally by more or less rapid gradations into clays, or in the form of irregular lenticular bodies of widely varying vertical and horizontal extent in the midst of the argillaceous layers. No regularity whatever exists in the position these bodies occupy in either the vertical or lateral range of the formation; they may or may not have intercommunication; they may be encountered at any point between base and summit of the shales; they frequently overlap.

The distribution of the materials here given for the oil-bearing zone of the Pierre, established both from well records and surface observations, is in harmony with nearly all the facts of occurrence of the oil, whether it is derived from several horizons in a single well or from but one point in depth, whether the yield be great or small at the same or different horizons, whether copious at one well, and at another within a comparatively short distance, the merest trace. In all, the yield is said to be from the more sandy variety of shale.

Thickness of the Oil-Bearing Zone.—This is not yet established. There are but few borings of a greater depth than 2000 feet below the surface. In two or three of these, small yields have been encountered, but the strata below the horizon assumed in the sections as the base of the zone, are still unexplored. Both the base and the summit of the zone in the sections have been graphically determined in accordance with the depths and locations of the shallowest and deepest producing horizons in any of the wells of the field. The thickness thus developed is 1947 feet.

The position of the zone thus defined in the stratigraphical series of the Montana beds it is impossible to determine absolutely, for, although in the Florence field the succession of strata from the summit of the series down is very probably normal, when the entire area underlain by the series throughout the West is considered, the

thickness of the group undergoes the greatest variation from point to point.

Extent of the Field.—It is obvious from the foregoing that the periphery of the productive portion of the Florence field may lie but a short distance within the area of outcrop of the oil-bearing zone as established in the sections. There is no geological evidence against such possibility; there are arguments in its favor, both from the known composition and distribution of the materials in the Pierre and from the structure within the region; there is also direct evidence favoring the extension suggested in the presence of oil in several wells drilled at various points considerably beyond the outskirts of the present field of development. As regards direction, extension to the east will be slight on account of the proximity of the area already producing to the eastern rim of the synclinal; to the west it will be limited by questions of economy and depth to which drilling can be advantageously carried; while to the north and south obstacles to extension within the limits defined are apparently wholly absent. In the extension of the field to the west and south, but particularly in the former direction, consideration should be given to the character of the oil-bearing bodies, that is, that they are masses of arenaceous materials inclosed in clay. Dip will therefore have no effect upon the flow of the oil beyond the body immediately affording the yield, unless two such bodies should be intercommunicating through the character of their materials.

The Source of the Oil.—The lowest geological horizon at which oil has been observed in Colorado, is near the base of the Jurassic shales. Although the wells affording this yield were sunk in the gravels of the Oil creek bottom, it is the statement of Mr. Johnson, of the Florence Oil Company, that they were derived from sandy beds in the shales themselves. The yield is reported as having been very small, 1 barrel per well per day.

Asphalt occurs in the Dakota in several localities in the West. There is an artificial spring at the base of the ridge near Turkey creek cañon, 15 miles southwest of Denver. The product may here be seen slowly oozing from the bedding planes of the sandstones in the upper portion of the formation. For a half mile along this portion of the hogback, the sandstones are discolored a dirty brownish black, and when broken yield with a greater or less degree of flexibility. In many localities in Wyoming the Dakota is pre-eminently an oil bearing formation, though as yet undeveloped.

The limestones and calcareous shales of the upper half of the

Benton formation are quite generally noted for the strong odor of petroleum that comes from freshly fractured faces or upon rubbing two pieces together. Slips, faces, lamination planes, and the contained casts of animal remains, particularly sharks' teeth, frequently have a coating of a bright, oily lustre. The beds possessing these characteristics are much blacker than the other portions of the formation.

The Niobrara only occasionally yields the odor of petroleum in a few of its beds, chiefly the basal limestones.

The Pierre has proved itself, at least for the Florence field, the great oil-producing formation. The oil cannot have been derived from the Jura, since the absorbent Dakota sandstones intervene; nor from the Dakota, since a coarse and porous sandstone would not yield any contained liquid to a body of shales; nor from the Benton, since the lower limestones of the Niobrara are too non-absorbent. It is, moreover, difficult to conceive a system of fissuring sufficiently extensive to have permitted the passage of oil from lower beds to the Pierre over such an areal extent, even upon the supposition of a more absorbent composition of the latter formation, and the fact that the synclinal structure would have been conducive to fissuring from below upward. The possibility of fissures both of an extent sufficient to have established communication between the formations below and the Pierre, and within the Pierre itself, has already been considered; but in this connection it may be further remarked that fissuring of the nature here suggested is beyond conception, for it could not have occurred in the Benton and overlying formations without occurring in the Dakota, in which case the great volume of water of the latter formation would have followed the oil and, upon exhaustion of this, appeared in the wells as an artesian flow. Not only is this not the case, but the majority of the wells remain dry after exhaustion, and none shows a water charged with salts peculiar to recognized Dakota flows. Moreover, the tendency of the Pierre shales, from their argillaceous composition, to close such fissures, has also been noted.

It would seem probable, therefore, that both source and reservoir of the oil lie within the Pierre formation itself,—a condition of affairs quite natural, since, on the one hand, shales are in many regions the recognized place of origin of oil, and, on the other hand, the distribution of the materials of the Pierre would permit the accumulation of the entire product of the formation in the more open, porous, sandy bodies inclosed within the argillaceous portions.

From the foregoing discussion it will be seen that no inferences can be drawn from the geology and well-data of the Florence field as to the existence of oil-bearing strata in other portions of the Rocky Mountain region and the adjacent prairies, even where underlain by beds of the same formation. That the Pierre carries oil in one locality has hardly the same force as an argument for its presence in another, as has the occurrence of workable coal in the Dakota formation, within a certain region, for its reappearance in another. In either case it is merely a recognized possibility that ought to be considered.

THE WELLS.

On account of the relations which the companies of the Florence oil-field hold to each other in extent of development, in capital invested, and in business competition, a detailed discussion of the wells cannot be presented in this paper, but the general deductions given are based upon a most careful scrutiny of the individual records of almost the entire number of wells—a scrutiny rendered possible only through the great courtesy of the several company managers.

Range of Flows in Stratigraphical Horizon.—Within a distance of 1155 feet beneath the summit of the oil-bearing zone, as provisionally established, no persistent succession of unproductive measures exists of a thickness greater than 100 feet. Midway between the depths of 1155 feet and 1947 feet,—the latter being the horizon of the lowest present yield and the arbitrary base of the oil zone,—there are three more productive layers within a distance of 100 feet of one another, separated from those above and the one below by approximately 350 feet of beds in which no flows have yet been struck. To obtain this close succession of productive strata it is necessary to consider the field in its entirety. Within the area of a particular group of wells the intervals barren of oil are often increased to from 200 feet to 400 feet, while in single wells they may become as great as 900 feet.

In the examination and tabulation of the well-records of the Florence field, two points of interest in connection with the flows were developed: the first, that for the field in general there are certain horizons in proximity to which oil has been found more frequently than in other parts of the vertical scale; the second, that wells yielding petroleum from depths of 400 feet or 500 feet below the summit of the oil-bearing zone, occur to the west of wells in which

no flows are encountered at these depths and that, too, notwithstanding the continuation of their productive horizons considerably beyond the easternmost wells before their dip brings them to the surface. In explanation of the latter point, it is possible that the requisite sedimentary conditions for an oil-bearing rock do not exist over this portion of the field, but from an inspection of the sections and from the fact that oil is rarely found at a depth less than 800 feet or 900 feet beneath the surface, it would seem that some other cause, such as evaporation, may have had an influence toward this result.

The vertical succession of oil-bearing strata in individual wells suggests another argument in favor of the origin of oil within the Pierre formation itself: in the event of its having been derived from a lower formation, it would have been largely taken up by the lower bodies of sands; the yields of the several horizons of a single well, however, afford no evidence of this condition, large flows occurring with equal frequency from either upper or lower oil-bearing bodies or from both.

Relations between Wells.—No connection exists for the field in general between flows of the same horizon. Only within the narrower group limits have they been recognized, and here, between wells in close proximity to one another. But a single instance is known of an extent of influence greater than 100 feet or 200 feet, when a newly drilled well caused a marked and rapid decrease in the yield of another from the same horizon 1000 feet to the north-east, the dip being west. On the contrary, wells have been sunk within 25 feet of another, without meeting a trace of oil at the horizon which in that first drilled afforded an excellent yield. In explanation of instances of this nature it is to be remarked that besides the normal rapid changes in texture and composition of the oil "sands," such bodies, themselves containing considerable clay and surrounded by argillaceous shales, may, in the course of compression and flexing which they have here and there undergone in earlier times, have been rendered locally more compact in texture or have become suddenly and confusedly blended with the mass of shales, thus presenting irregularities in arrangement of materials, all with influence upon the flow of the oil that may be contained within the limits of disturbance. Instances of this nature might also be accounted for by fissures, provided that direct evidence of their existence was found at the time of drilling.

Percentage of Successful Wells.—Within the area of the oil-bearing zone of the Pierre formation, there are accounted for as having been drilled up to November 1, 1890, 82 productive and unproductive wells. Of these, 16 per cent. have shown no trace of oil, and 19½ per cent. have shown traces less than an appreciable flow in from 1 to 6 horizons in a single well, or, in all, 35½ per cent. unproductive wells. If to this be added 7 per cent.—the proportion of wells yielding less than 5 barrels and consequently below the present standard of economic pumping—the total percentage of unproductive wells will be raised to 42½, leaving 57½ as the percentage of successful wells.

The Yield.—When oil is struck in the Florence field, nearly all the productive wells show an abnormal rise, due to expansion of the gas contained in the oil; the oil ascends in the well from 50 to 1000 feet, but upon being pumped settles within a few days to a level a little above that of the flow—a level which is maintained until the well clogs or exhaustion begins to assert itself. The original yield of the wells, upon attaining the normal level, has ranged from 5 to 150 barrels per day; several flows have yielded between 75 and 150 barrels, and a large number from 25 to 75 barrels. After a longer or shorter time a gradual and uniform decrease in production sets in, which, however, may extend over a period of several years before exhaustion is carried to the limit of economic pumping. Of the smaller wells, many may be temporarily exhausted by forced pumping, but after the lapse of a short interval of rest completely regain their former yield. Among the wells there are some that have maintained a constant flow during their entire existence to date—several years—while in a few instances actual increase to two or three times the original yield has occured. Whenever a yield has diminished through clogging of the well by accumulation of extraneous material, cleaning rarely fails to restore it to its normal amount.

The areal distribution of the oil, as evidenced by the yield of widely separated wells, is exceedingly uniform for the portion of the field thus far developed, both large and small flows being alike encountered in each of the various well-groups.

The use of torpedoes has been tried in the Florence field in three instances, without success. Caving of the walls has resulted in each case, while within the range of action of the torpedo it is believed that on account of the clayey nature of the strata there has resulted a packing of materials, the reverse of the desired effect.

The life of the wells of the Florence field is very variable: there are some of an age of over four years which are still good producers; many, over two, to date, among which are several of the largest; while of those whose duration has been less than a year there are comparatively few. The well of shortest life was one of six days of a total yield of 37 barrels. Examination of well-records fails to disclose any relationship between the life and the size of the flow.

Gas.—In the Florence field gas occurs chiefly in association with the oil, although small amounts have been encountered in the shales at horizons where no trace of oil was found. Associated with the oil, its quantity and rate of delivery is very variable. In some wells it is quickly exhausted while in others it remains persistent with a fair flow. There are instances in which it has manifested tremendous force upon being struck, carrying the oil beyond the derrick tops and for a while rendering difficult its control, afterwards quickly diminishing to an inappreciable amount. In other instances, after the first rush of gas, its flow has been maintained at a constant pressure and delivery during the entire life of the well. Neither pressure nor flow have ever been measured, but they are slight in comparison with the eastern fields. The gas has been utilized only in lighting the immediate vicinity of the wells.

Water.—This is met with in a few wells but it is an uncertain element. Its quantity is always small. Except in two instances it is fresh and has been found within 350 feet of the surface. The exceptions are salt, and were found in connection with the oil at a considerable depth. None of the water is artesian. No relation exists between the water and oil in the Florence field in the lines usually attributed to them in eastern wells. Indeed, many of the wells are not only without any trace of water from the surface down but remain dry after exhaustion of the oil. There is nowhere any evidence of a water-pressure back of the oil.

THE OIL.

Character, Production, Treatment, Products.—The crude product of the Florence field is of a light yellowish-green color, varies but slightly from an average gravity of 31° B., and is remarkably free from sand and other impurities, which in a month's constant run amount to but about a gallon per still.

The present production of the field is about 2000 barrels per

day, yielding upon distillation approximately one-third refined or illuminating-oil, a small amount only of lubricating-oil, and a residuum, which is stored. The production, however, is varied according to the demand for oil. The capacity of the field is considerably above 2000 barrels daily. For the twelve months to October 1, 1891, the total product amounted to about 300,000 barrels of crude oil, from which was manufactured about 100,000 barrels of illuminating oil and 5000 barrels of lubricating oil.

The treatment of the crude oil as practiced by the United Oil Company, in the process of distillation, is the same as ordinarily employed elsewhere. It passes from the wells to storage-tanks, and thence to the stills in charges of 400 barrels for the larger and 100 barrels for the smaller. The run of the former is of 36 hours' duration, of the latter 20 hours. The yield from these is 40 per cent. illuminating-oil, the gravity of the distillate being 44½° B., of the residuum, 22° B. From the stills the cooled distillate passes to the distillate storage-tanks and thence to the agitators, where it is subjected to chemical treatment consisting of agitation with sulphuric acid (66° B.), and subsequent washing with a solution of caustic soda and water. Upon completion of the agitation it is run to the bleachers, settled, and from there distributed to the warehouse for packing or to the tank cars direct. The percentage of naphtha is insignificant. The burning-point is 130° F.

The lubricating oil is derived from the residuum by passing the latter through a retort which removes the paraffine and raises the fire-test. This "stock" is compounded with neutral oil from the stills, which results in a product having a cold-test from 0° to 30° F., a flash-test of 350° F. and a gravity from 24° to 28° B. The residuum chills at 60° F.

The quality of both illuminating and lubricating oils derived from Florence crude petroleum is said to be excellent; but for the latter product there is at present but slight demand, owing to eastern competition and freight-rates.

The residuum from the manufacture of the above two classes of oil is stored in open reservoirs about the field, except such portion of it as is required beneath the stills and boilers of the refineries and wells. In this use of it from 1200 to 1300 pounds (three to four barrels), are estimated to be equivalent in effect to 2000 pounds of coal, worth from $1.43 to $2 per ton at the mines. It is thrown into the fire-boxes in jets with the aid of super-heated steam, and not

only affords a fire capable of instant regulation, but is of great economy in the saving of labor.

The tanks for storage of the various products of the Florence field range in size from 5000 to 35,000 barrels.

Discussion.

In reply to the question, whether his investigations in the Florence field had given any evidence concerning the origin of the petroleum, as animal or vegetable, Mr. Eldridge said:

I omitted that topic from my paper, because I could throw no light upon it. The oil has not a strong odor. The formation contains animal remains in abundance, and through the fossiliferous lime-concretions there are many minute traces of plant-life, such as stems or particles of bark. As seems to be the case in most oil-bearing formations, nearly all evidence of life, except of animal life, has disappeared. The formation contains no coal in this locality, but coal is found in it at several localities in the West.

F. H. Newell, Washington, D. C.: Mr. Eldridge's paper throws some light on the subject of prospecting for oil. In the western fields we have had a curious succession of discoveries of oil. In Pennsylvania, West Virginia, and Ohio various theories have been projected or deduced, confining the occurrence of oil to certain conditions of structure, texture, position, etc.; but, on the other hand, the oil-operator himself, who backs up his opinion with his money, has treated with disregard, and even with contempt, all such studied generalizations, and has drilled his wells anywhere that fancy dictated. The result has been that he has spent immense sums, but he has discovered oil where the best experience would not have led those who knew most about it to advise him to drill. In the Florence field we see the same thing. Here is a formation, the structure of which is quite the reverse of anything we had previously recognized as favorable, and near the upper limit of which, strange to say, oil has been found, nevertheless. Without discrediting the general conclusions which very able experts may have drawn, this case shows that in trying to anticipate the discovery of oil we are still completely at a loss. The "oil-man," with his persistence in drilling where he ought not to drill, has often brought valuable resources to light.

THE BENDIGO GOLD-FIELD.

BY T. A. RICKARD, ALLEMONT, ISÈRE, FRANCE.

(Glen Summit Meeting, October, 1891.)

AMONG the names which won a world-wide fame during the golden age of the early fifties, Bendigo and Ballarat were to Australia, what the Yuba and Grass Valley were to California. The map of Victoria did not for a long time show the name of old Bendigo;* for this first and more distinctive name was replaced with the more English Sandhurst, just as the alluvial diggings gave place to quartz mining. Towards the close of last year, steps were taken to give back the old name, as associated with the early days of rich alluvium, and more suggestive than the application taken second-hand from an English military academy.

HISTORY.

The first discovery of gold† in this district, was made in the autumn of 1851; but there has never been any certainty as to the day or the man. At that time the country around Bendigo Creek was a part of the Ravenswood sheep-run, and its resemblance to the Forest Creek district (now Castlemaine), induced the first prospecting. It was late in November when the "rush" broke out; the shepherds left their flocks; the sylvan solitudes were disturbed by

* The name of Bendigo is said to have been derived from a hut-keeper on the Ravenswood sheep-run, who on account of his fondness for "fisticuffs" was nicknamed Bendigo, after the prize-fighter of that name. It is not aboriginal, as is often supposed, but Spanish; and equivalent to our Benedict. There must always be some confusion between the two names of Sandhurst and Bendigo; the town, and with it the gold-field, having been three times named. Several other old familiar names have been likewise unfortunately replaced by second-hand English ones; so that an old digger talks of Forest Creek when he means the modern Castlemaine, Mt. Ida for Heathcote, Growler's Creek for Bright, etc. I shall use the names Sandhurst and Bendigo interchangeably.

† The first discovery of gold in Australia, was made by E. H. Hargraves, February 12, 1851, near Bathurst in New South Wales. In August of the same year the discovery at Buninyong, near Ballarat, inaugurated the first of a series of rushes to the Victoria gold-fields.

the voices of the first "diggers;" the green glades of Bendigo Creek were ruthlessly uptorn by pick and spade; and the busy energetic life of a young and thriving mining camp replaced the sleepy idle routine of an out-of-the-way sheep-station.

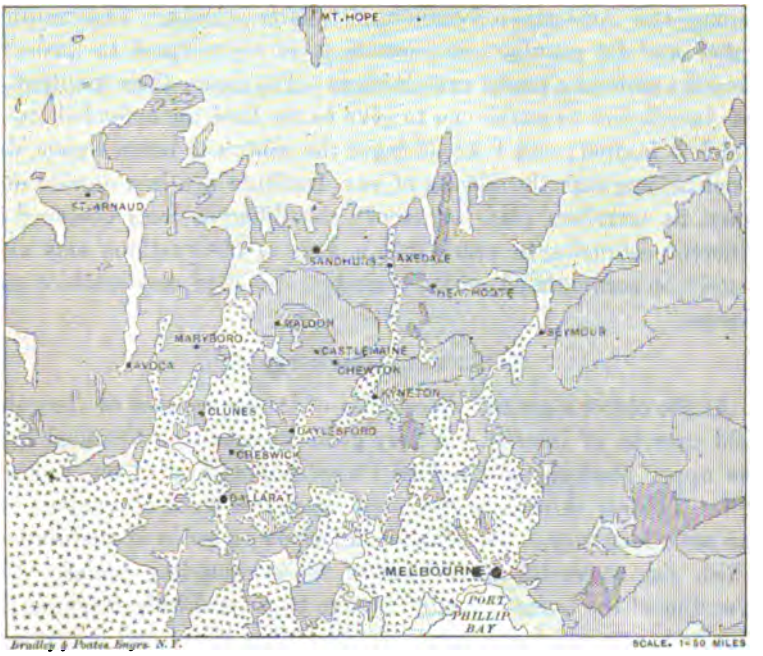
Since that date the gold-field has yielded over fifteen million ounces of the precious metal, valued at more than sixty millions sterling. Like most mining districts, it has passed through days of severe depression and extreme inflation. Until 1854 the alluvium only was worked. The pan and the cradle early gave way to the puddling-tub, a machine more suited to the clayey character of the wash. By reason of the limited supply of water, the long tom was never very widely used. The highest output on record was reached in 1853, when 661,729 ounces, valued at £2,646,800, were obtained from the alluvium only. Long before the diggers were aware of the real value of gold-quartz, they used to amuse themselves by breaking off specimens from the outcrops for decorative purposes.* Diamond Hill owes its name to the beautiful specimens which its surface yielded in the early days. Vein-mining or "quartz-reefing" had its inception with the discoveries made by Ballestedt on Victoria Hill, which has now been pierced for a depth of over half a mile, and is honeycombed with deep workings. From 1854 to 1862, the quartz-mining industry had a chequered career; as will be readily understood when the peculiar character of the ore-deposits has been passed in review. In 1859 the first regular registration of a quartz-mine† took place. About this period the shallow alluvial deposits became exhausted and, in the absence of deeper channels, the whole energy of a very enterprising community was concentrated upon the exploitation of the quartz-lodes. In 1862 the first limited-liability company was brought out; and in the immediately succeeding years the steady development of the field and the increasing knowledge of the lode-structure were accompanied by an increasing gold-output, which rose in 1870 to 241,380 ounces. The formation of numerous companies which marked the early seventies, introduced fresh capital into the district, and thus led to great increase in the work done in the mines, the output from which averaged at this time over 300,000 ounces per annum. But company-promotion soon devel-

* So ignorant, it is reported, were the pioneers of gold-mining in those days, that many immigrants went to the rushes or mining stampedes with the idea that "gold in quartz" meant gold in *quarts* or pailfuls!

† This was the Johnson's gold-mine, which is still a rich producer, having headed the dividend list for 1890 with £30,800.

oped into a mania which rapidly did its evil work. Paper mining replaced honest work, as it has done in many another mining district before and since. A reaction set in; the gold-field saw its darkest days; and a collapse took place which ended the wild speculation in bogus companies and marked the commencement of a new era of steady progress. The opening up of deeper ground and a renewal of

Fig. I.



GEOLOGICAL SKETCH MAP
OF A PORTION OF
VICTORIA.

rich discoveries, brought about a revival in the first years of the last decade, which, accompanied as it has been by great enterprise, and the intelligent development of the mines with a more accurate knowledge of their ore-deposits, has enabled the Bendigo gold-field to hold the first place among the quartz-mining districts of Australasia, and to make its record unique in the statistics of this industry.

To the members of the Institute, probably few mining districts are less known, considering their importance, than those of the colony of Victoria. The steamship, the railway, and the telegraph, have linked together the most distant mining centers, Broken Hill and Virginia City, Johannesburg and Grass Valley; but the older gold-fields of Australia have received but a superficial and passing notice from the pens of standard writers. More particularly is this true of Sandhurst or Bendigo,* which shares with Ballarat the first rank among the Australian localities of quartz-mining. This mining center and its peculiar ore-deposits it is my purpose to describe, from the notes of a recent examination. The time—three months†—was insufficient to enable me to give to the field the extended study which it invited; and I would leave the subject to better hands, did I not believe that the objects of our Institute are best carried out, when its members place on record the different facts observed in different countries, in order that a mass of observations may ultimately be accumulated, which those best qualified may combine and discuss.

General Description.

At the outset a general description of the appearance of the gold-field may be of interest. A very good view can be obtained from the upper platform of the Old Chum poppet-heads,‡ on Victoria hill. The air is usually clear, and from this point of vantage one can see a long way. The district lies among a series of undulations which rise above the tame level of the surrounding plains. The even line of the latter is further broken by several hills, marking the granite bosses which penetrate the overlying slates and sandstones. To the north, poppet-heads and tall red brick chimneys in long succession indicate the various great lodes or "lines of reef,"§ stretching out to Eaglehawk, four miles away, while out in the distance be-

* Only the more recent works on gold-deposits contain any references, and these are almost invariably inaccurate and misleading in their descriptions of the mode of occurrence of the quartz. Lock's *Gold* which is, generally speaking, a very complete compilation on the subject, contains only one reference to Sandhurst, comprised within a paragraph of a dozen lines.

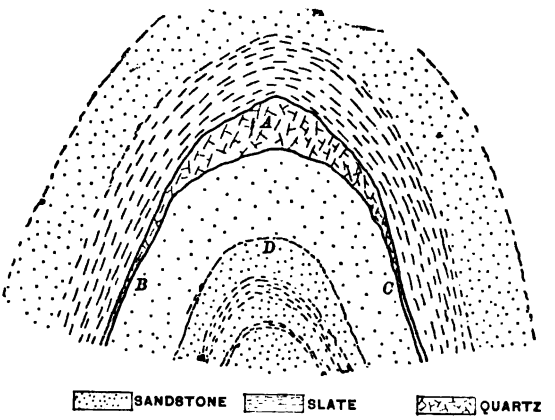
† April, September, and part of October, 1890, also a part of February, 1891.

‡ "Poppet-heads" is the English and colonial equivalent for the Western "gallas" or gallow's frame. "Sheer-legs" and "heapsteads" are the names used in the north of England.

§ A "line of reef" is a reef or lode taken as a whole. The mines of Amador, Calaveras, and Tuolumne counties would be said to be on "the same line of reef," viz., "the Mother-lode" of California.

yond is the dark blue sea of the trackless bush,* broken in front by the hills at Kerang and to the right by the promontory of the White hills,† a name familiar to every Australian digger. White hills they are no longer, for the "cement" which covered the rich alluvium has become oxidized, and patches of red gravel-heaps have replaced what was once glistening white. Between these and the spectator lies the modern city of Sandhurst, its well laid out streets lined with the English trees which have replaced the gums that once covered the site. A wavering, blue, irregular line marks Bendigo creek, once flowing through the forest glades, but now meandering past tailing-heaps and back yards. Out beyond the town we

Fig. 2.



SADDLE.

can dimly see Mt. Ida, where Heathcote lies. Southward the succession of poppet-heads marks the auriferous belt. The engine-houses of the South Bellevue and Eureka Extended mines indicate the New Chum reef, on which there is operated to-day perhaps the greatest series of deep gold-quartz mines which the world has yet seen. A new church rises above Golden Square, the scene of some of the first and richest of the alluvial diggings. The view is further diversified by gray heaps of tailings, bluish piles of waste rock, gar-

* The forest of *Eucalyptus*, which at one time covered the whole of the habitable portion of the Australian continent.

† The White Hills of Bendigo and the Black Hill at Ballarat are in the history of Australian gold discovery what Mokelumne Hill in old Calaveras, and Table Mountain in Tuolumne were to California, or what Ruby Hill at Eureka, and Mt. Davidson at the Comstock were to Nevada.

dens and houses, among which is seen the sinuous curve of the railway along which the Melbourne express is now coming. Further south, in the distance, are the granitic slopes of Mt. Alexander, beyond which lies Castlemaine. To the west, clouds of sulphurous smoke indicate the various pyrites-works, and a black line against the blue sky marks the water-supply flume, while immediately below us are the gardens and house of one of the mine-owners,* an oasis of pleasant green among the stern practical surroundings of mines and mills.

Sandhurst or Bendigo has a population of over 30,000 inhabitants, several fine public buildings, and some very beautiful public gardens. It is connected with Melbourne, the chief port and metropolis of Victoria, by a double-track railway 101 miles long, constructed at a cost of £18,000 per mile.

Statistics of Production.

A few figures, taken from the annual report of the Secretary for Mines, will indicate the size and importance of the Bendigo gold-field. During the past two years the gold-production of the three leading Australian colonies has been as follows:

	Ounces.	Value. £.	No. of miners.	Average per miner. £.	s.	d.
Victoria, 1889, . .	614,838	2,459,352	24,323	101	2	2
" 1890, . .	588,560	2,354,240	23,833	98	15	7
New South Wales, 1889,	119,758	434,070	10,192	42	11	9
" " 1890,	127,460	459,086	12,182	37	13	8
Queensland, 1889, .	737,822	2,582,377	8,955	288	7	5
" 1890, .	600,000	2,100,000	.....	.....		

The year 1889 was the first in which Victoria did not lead the colonies, being surpassed by Queensland through the Mt. Morgan mine, whose output alone was over 300,000 ounces. The marked decrease in the yield of Queensland for 1890 is similarly due to a decline in the output from Mt. Morgan. In like manner, the Queensland average of £288 per miner, in 1889, is explained by the fact that the most of the value of the product of Mt. Morgan in that year found its way into the pockets of half a dozen men.

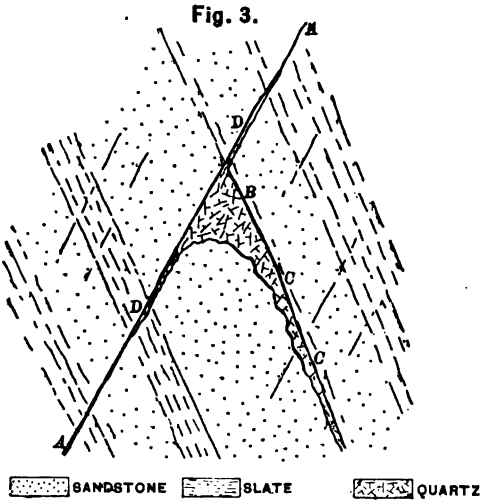
The yield of 588,560 ounces (1890) is the lowest on record in the history of gold-mining in Victoria. There has been a gradual decrease during the past decade, mainly due to the exhaustion of the more readily accessible alluvial deposits.

* Fortuna Villa, the residence of Mr. George Lansell.

From 1851 to the end of 1890 the yield of gold from Victoria amounted to 56,870,574 ounces, valued at £227,482,296, an average per year of 1,452,761 ounces, the highest for any single year being 3,053,744 ounces, in 1856. Of this total, Bendigo has contributed 11,168,414 ounces, valued at £44,673,656. In addition, it is estimated that the gold taken away privately to Melbourne and the neighboring colonies, and not included in the government returns for the district, will amount to 4,000,000 ounces, which would give a round total of over 15,000,000 ounces, valued at over £60,000,000 sterling.

Victoria is divided into seven mining districts, of which the

Fig. 3.



FALSE SADDLE.

most important are Ballarat and Bendigo, the former being the chief alluvial center, while the latter leads the quartz-mining districts. The total area of the Victoria gold-fields, 86,760 square miles, is worked and prospected by a force of 23,833 miners; and the proportion belonging to the two principal gold-fields is as follows:

	Area in sq. miles.	Miners.
Bendigo,	5,870	4,420
Ballarat,	5,180	6,249

While the mining operations in the Ballarat district are distributed over an actual area of 40 square miles, and the boundaries include such important centers as Clunes and Creswick, those of the Ben-

digo district are practically concentrated upon an area of 21 square miles within the adjacent townships of Eaglehawk and Sandhurst. During the past 2 years the yield of gold has been:

	Alluvial.			Quartz.			Total.		
	Oz.	dwts.	grs.	Oz.	dwts.	grs.	Oz.	dwts.	grs.
Bendigo, 1889,	6,973	12	10	134,547	8	21	141,521	1	7
" 1890,	3,293	3	18	134,671	10	11	137,964	14	5
Ballarat, 1889,	98,342	6	13	117,321	18	0	215,664	4	13
" 1890,	92,836	2	10	117,597	0	8	210,433	2	18

It is seen that, while Ballarat is the leading alluvial district, it produces also a large proportion of gold from quartz,* while at Bendigo, on the other hand, the alluvium is relatively unimportant. The figures just given bear out the statement of the Secretary for Mines that the diminution of the gold-yield is chiefly in that from the alluvial mines. In both the leading centers the yield from quartz has slightly increased, while that from alluvium has considerably diminished. The average yield of the quartz per ton of ore crushed, during the past year, was as follows:

	dwts.	grs.
Victoria (average of the seven districts), ...	9	4
Bendigo, ...	9	5
Ballarat, ...	7	21

The average yield from pyrites and blanketings was:

	Total treated.		Total yield.			Average.		
	Tons.	cwt.	Oz.	dwts.	grs.	Oz.	dwts.	grs.
Bendigo, . .	1,766	10	3,901	16	12	2	4	4
Ballarat, . .	2,148	0	5,196	19	7	2	8	9

The highest and lowest prices paid for the gold per oz. were, at Bendigo, £3 17*s.* and £3 19*s.*, and at Ballarat, £3 17*s.* 6*d.* and £4 3*s.*

While in the colony as a whole the number of miners is pretty equally distributed between alluvial and quartz mines—11,470 in the former to 12,363 in the latter—there is, in the two leading districts, a greater disproportion; Bendigo having 949 alluvial and 3375 quartz-miners, while Ballarat has 2440 alluvial and 3677 quartz-miners. At Sandhurst there are 270 Chinamen, nearly all engaged in surface-washing, while at Ballarat there are 736 Mongolians to 5508 Europeans.

* At Ballarat, as the deep leads (alluvial) were becoming worked out, the croppings of the quartz-veins were found in the bed-rock of the alluvium. Several claims which have been rich in alluvium are now good quartz-mines.

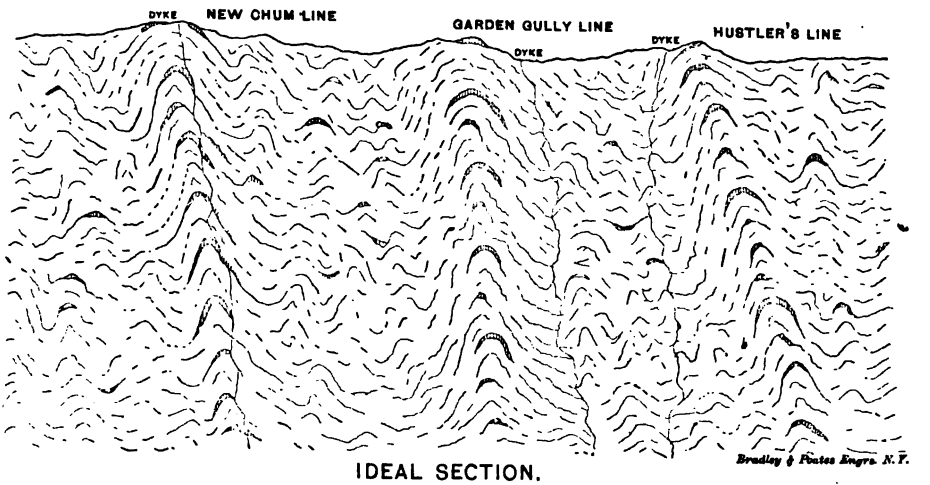
The Bendigo gold-field includes several scattered subdivisions; but it consists practically of Eaglehawk and Sandhurst (or Bendigo), two distinct municipalities, forming one long straggling township, somewhat after the manner* of Gold Hill and Virginia City, or Blackhawk and Central City.

In 1890 the machinery employed in quartz-mining was thus distributed:

	Steam-engines.	Stamps.	Machine drills.	Concentrators.	Arrastras.
Sandhurst, . . .	175	625	70	65	25
Eaglehawk, . . .	105	501	40	40	6

For the Bendigo gold-field as a whole, the numbers are as fol-

Fig. 4.



IDEAL SECTION.
SHOWING THE MAIN LINES OF REEF.

lows: Steam engines, 316; aggregate nominal H.P., 6873; stamp-heads, 1328; whims, 76; whips, 127; machine-drills, 128; concentrators, 105; arrastras, 82. The total value is estimated at £478,611. Considering the size and yield of the district, the value of the machinery is small indeed, when compared to that of a Western American mining-camp, with its much larger and more costly hoisting-engines, mills, etc.

* But the mining townships of Victoria are very different from those of the great West. Ballarat, Sandhurst, Clunes, and Creswick are pretty towns, situated in rich agricultural districts, and presenting none of that bare ruggedness which is characteristic of most mining camps of Colorado, Nevada, and California.

The quantity and cost of the timber consumed for mining purposes during the year were as follows:

Firewood,*	146,628 tons.
Props and cap-pieces,	105,533 pieces.
Laths and slabs,	80,820 "
Sawn timber,	821,232 feet.

The value of this is set down at £63,856 6*s.* 6*d.*

Mining is carried on for the most part by companies, whose share capital is invariably small and their reserve capital usually *nil.* The tribute-system is largely in use, and, here as elsewhere, often leads to the discovery of valuable ore-bodies. There are some private mines, the most important of which belong to Mr. Lansell, a mine-owner, the record of whose enterprise forms an important part of the history of the field. A few small parties of miners, less numerous than the future prosperity of the district requires, are engaged in prospecting new ground.

Operations are distributed among eleven approximately parallel formations or "lines of reef," of which by far the most important are the New Chum,† the Garden Gully and the Hustlers. At present the greatest activity prevails along the first-named which has also been the most continuously productive of the series. A notion of the extent of the field and its worthiness to rank among the greatest of modern mining centers may be obtained from the following statements. The New Chum lode has been worked from Axe Creek to the Franklin mine in Sailors' Gully, a distance of fourteen miles. The Garden Gully has been followed from the Suffolk Tribute and the Moon, beyond Eaglehawk, to the Great Southern Extended, beyond Bendigo Creek, a distance of 7 miles. The third great lode, the Hustlers, has been tapped at the Fortuna Hustlers in the city of Sandhurst, and from there has been worked as far as the King of Prussia, in Opossum Gully, which is 5 miles away. So much for the extent of working along the strike. Downwards, the New Chum has been followed for a vertical depth of over half a mile (2641 feet), the Garden Gully for over 2300 feet and the Hustlers 2000 feet. There are in Sandhurst 18 shafts exceeding in depth 2000 feet. Several of these are still going down; and of the

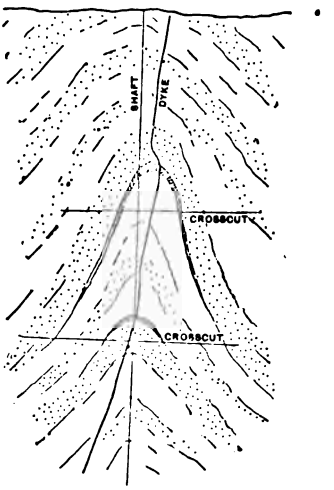
* Fifty feet of firewood equal one ton of 2240 pounds. Laths and slabs are the same as what the American calls "lagging."

† "New Chum" is the colonial equivalent of the Western "tenderfoot," i.e., a fresh arrival in the country.

20 companies among which these 18 shafts are distributed, 3 are in rich ore, 7 are breaking pay-ore, 9 are prospecting and 1 is idle.

Though the output has suffered during the past two years by reason of the greater attractions presented by gambling in silver shares at Broken Hill, Sandhurst contained during the past year 28 dividend-paying companies which produced 101,879 ounces 10 dwts. (not including 7 tribute parties which were dividend paying), enabling the payment of £149,381 17*s.* in dividends. The work of the past year has shown an improvement as indicated by the following comparison. In 1889 the calls amounted to £137,489, the

Fig. 5.



dividends to £118,473, a balance on the wrong side of £19,016. In 1890 the calls were diminished to £111,142, while the dividends increased to £149,381, leaving a profit of £38,239. During the last 14 years the dividends have exceeded the calls by £1,101,836, the year 1889 being the only unprofitable period. Roughly speaking, during 1890 each ounce of gold was obtained at a cost of £2 10*s.* 8*d.*, leaving £1 9*s.* 4*d.* as profit for the shareholders.

Out of the 28 companies which appeared on the dividend-list at the beginning of 1891, the 20 whose records are obtainable show that only four have failed to pay back the capital expended upon them, and of these four, one is not yet a year old. The

average total dividends of each of these 20 properties* amounts to £81,947, the average nominal capital of each is £49,742, while the average paid up capital is only £31,167. It is seen, therefore, that these 20 companies have returned in dividends nearly twice the amount of capital called up for their equipment and development. Further, it should be stated that the number under consideration does not include several of the mines† which have the

* The following list gives the detailed figures of the twenty companies which were dividend-paying at the beginning of this year (1891). The results are given up to the end of 1890:

Company.	Nominal capital. £.	Paid up capital. £.	Dividends. £.
Catherine Reef United,	135,200	82,810	50,755
Fortuna Hustlers,	14,000	3,500	2,800
Gt. Extended Hustlers,	68,000	62,050	419,200
Hercules and Energetic,	60,000	8,250	74,625
Johnson's Reef,	78,000	72,100	223,950
Lazarus New Chum,	67,500	63,187	67,500
Lazarus No. 1,	67,500	63,950	95,062
Lady Barkly,	24,000	17,400	58,315
New Chum Con,	42,000	18,200	132,300
New Chum Railway,	36,890	28,589	52,078
New Chum United,	14,750	8,850	66,375
North Old Chum,	54,000	33,075	86,495
Rose of Denmark,	24,000	4,800	63,600
Shamrock,	45,000	27,750	15,750
Shenandoah,	96,000	33,600	59,600
South New Chum,	32,000	21,735	800
Specimen Hill United,	20,000	12,000	23,500
United Hustlers and Redan,	48,000	19,200	114,000
United Devonshire,	28,000	9,800	21,700
Young Chum,	40,000	32,500	10,550
Totals,	994,840	623,346	1,638,955
Averages,	49,742	31,167	81,947

† The greatest producers have been the following mines:

Name.	Called up capital. £.	Dividends. £.
Garden Gully United,	21,642	667,796
Gt. Extended Hustlers,	62,050	419,200
Johnson's Reef,	72,100	223,950
Gt. Hustler's Tribute,	61,200	620,200
North Johnson's,	31,850	148,625
United Devonshire,	8,244	224,000
New Chum Con.,	18,200	132,300

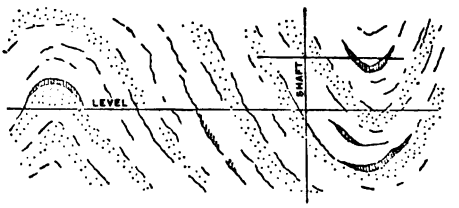
There are many others, the totals of whose dividends is more than £100,000.

greatest records, but have temporarily dropped out of the dividend-list.

The instances of individual productiveness here have been surpassed in other parts of the world; but it is doubtful whether in the history of gold-mining there can be shown a better record in the proportion of dividends paid to capital expended. The product of the mines from tribute-parties is not included in the above list unless especially mentioned. It is a matter of record, for instance, that, including the product of the tributers, the Garden Gully United has yielded over £1,000,000. The Kentish mine of the late J. B. Watson is said to have yielded over £2,500,000; and another private mine, G. Lansell's "180," has given magnificent profits to its owner, as have also several tribute-workings, the records of which are not accessible now.

To the Australian the name of Sandhurst is always associated with

Fig. 6.



INVERTED SADDLES.

the "saddle-reefs." It was the frequent mention of these which led me to visit the mines; and it is the peculiar interest which they must have for all geologists and mining engineers which has induced the publication of these notes. Before discussing, however, these and other matters of detail, a general account of the geology of the district should be given.

Geology.

As will be seen by reference to the map* of Victoria (Fig. 1, p. 465), the gold-field is situated near the northern edges of the exposure of the Lower Silurian, which, a short distance further north, is overlain by Pliocene shales. The boss of granite at Mt. Hope suggests

* The geological map of a portion of Victoria which accompanies this paper, was copied by me from the geological sketch-map in *The Geology and Physical Geography of Victoria*, by R. A. F. Murray, the geologist of the Victorian government.

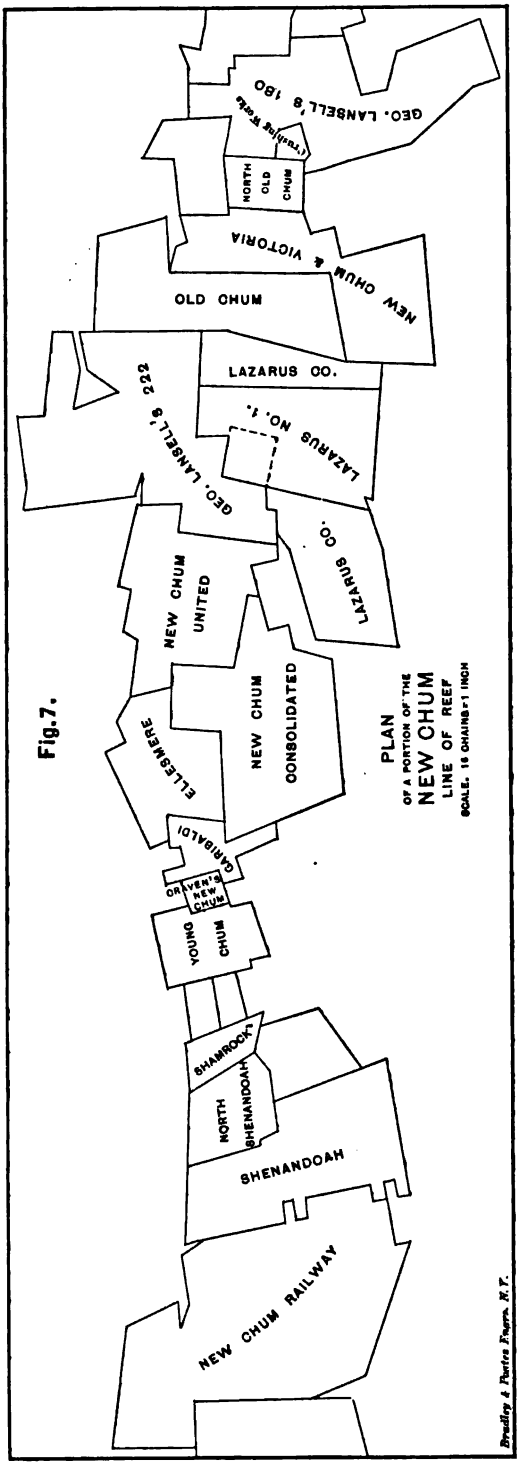
the deeper masses of crystalline rocks which also form the Mt. Alexander ranges, dividing the Sandhurst and Castlemaine gold-fields. The gold-mining districts of Victoria are almost entirely confined to beds of Upper and Lower Silurian age, of which Mr. Murray says:* "As surface or underlying rocks, they occupy the greater part of Victoria from the sea-coast to elevations exceeding 6000 feet." They form the bed-rock of the alluvium the yield of which astonished the world in 1851, and the country-rock of the quartz-lodes from which that alluvial gold was derived. Selwyn† computed their total thickness to be not less than 35,000 feet. While the line of division between the two horizons has not been found, certain differences in lithological character and fossil remains have led the Victorian Geological Survey to refer to the Upper Silurian that portion of the Silurian rocks lying east of a line drawn from Melbourne to Heathcote, while the Silurian west of that line is regarded as Lower. It is in the lower horizon, which, according to Professor McCoy, corresponds to the Llandeilo Flags and Bala rocks of Wales, that the auriferous deposits of Ballarat, Castlemaine, and Sandhurst‡ occur. In the upper are the Ovens district, the Buckland and Harrietville. Again, to the latter belong the picturesque alpine districts, and to the former, the extensive plains diversified by low rounded hills.

While there is a general similiarity in the mode of occurrence of the gold, there are also some interesting differences in the habits of the quartz-lodes in the two horizons. The auriferous deposits are found traversing certain defined belts, which have a general strike 20° to 30° west of north. These parallel belts contain veins of quartz which conform to the general strike, and which, like the belts themselves, are separated by barren portions of country. My observations lead me to believe that the gold-veins of the Lower Silurian are more often conformable to the stratification, while those of the Upper are more frequently true fissures, traversing the country at an angle to the bedding. R. A. F. Murray, the government geologist, has noted that while the quartz-reefs in the Upper Silurian

* Page 33 of the book mentioned in the preceding note.

† A. R. C. Selwyn, formerly head of the Geological Survey of Victoria, and now occupying a similar position under the Canadian government.

‡ The fossils most common in the rocks at Sandhurst, are graptolites, particularly *stellatus*, *extensus* and *tripedes*, together with *Sertularia magna* and *vergata*, *Didymograpsus fruticosus*, and *Phyllograpsus folium*. In the weathered slate-beds they are easily found, but underground the dark color of the rock, and the development of fine cleavage renders them difficult of recognition.


GEO. LANGELL'S 180
(Township of Eaglehawk)
NORTH OLD CHUM
VICTORIA & NEW CHUM
OLD CHUM
LAZARUS CO.
LAZARUS NO. 1.
GEO. LANGELL'S 222
NEW CHUM UNITED
LAZARUS CO.
ELLESMERE
NEW CHUM CONSOLIDATED
GARIBALDI
OR RAVEN'S NEW CHUM
YOUNG CHUM
SHAMROCK
NORTH SHENANDOAH
SHENANDOAH
NEW CHUM RAILWAY
Fig.7.
PLAN
OF A PORTION OF THE
NEW CHUM
LINE OF REEF
SCALE. 15 CHAINS=1 INCH
Bradley & Porter Fecit. N.Y.

may be fewer in number and smaller in size, they have, on the other hand, considerably exceeded hitherto, in their average yield of gold, the reefs in the lower division.* It seems probable, however, that the smaller number of discovered reefs in the upper horizon is largely due to the more mountainous and less accessible character of the country in which they occur. In the few districts, such as Bright and the Buckland, where there has been any considerable amount of prospecting in the rocks of this division, large numbers of parallel veins have been exposed. Their greater yield per ton is accounted for by the fact that the ores of the mines in the Upper Silurian are frequently so charged with sulphides as to come under the "refractory" class, necessitating the working of ore of higher tenor than is required in the Lower Silurian, where the quartz is usually of the simplest "free-milling" type. This difference will also serve to explain the circumstance, mentioned by the government geologist, that all the largest nuggets obtained in the alluvial mines of Victoria have been found where Lower Silurian rocks prevail.

The Bendigo district consists, broadly speaking, of a belt of sandstones and slates of Lower Silurian age, abutting to the south and west against the granite of the Mt. Alexander ranges and overlain to the north and northeast by the shales of the Pliocene. The most marked characteristic of these sandstone and slate beds is the extreme bending, folding, and contortion which they have undergone, accompanied, as might be expected, by a varying amount of fissuring and faulting. The anticlinal and synclinal undulations are often remarkably sharp and exhibit every gradation in extent, from a few feet to miles, from a hand-specimen to those larger corrugations of the earth's crust which geologists name "ge-synclines" and "ge-anticlines." The main anticlinal axes strike N. N. W. and S. S. E., but there are also transverse undulations which further complicate the geological structure.

The Reefs.

The quartz reefs or lodes† conform to these folds in the country-

* The returns show that the average yield per ton is as follows:
Upper Silurian, Beechworth, 13 dwts. 23 grs., Gippsland 15 dwts. 21 grs.
Lower Silurian, Ballarat, 7 dwts. 21 grs., Bendigo, 9 dwts. 5 grs.

† The Australian calls a quartz lode or vein a "reef," the Californian a "ledge," while in Colorado the word "crevice" often does similar duty. Throughout this article I shall often use the colonial mining terms, since they are usually expressive and it would be difficult to find other names that are not also local.

Fig. 8.

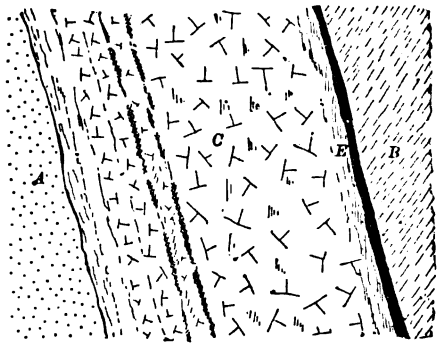


Fig. 9.

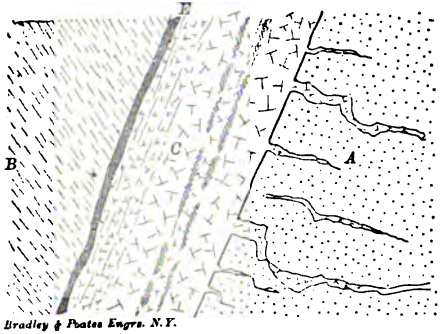
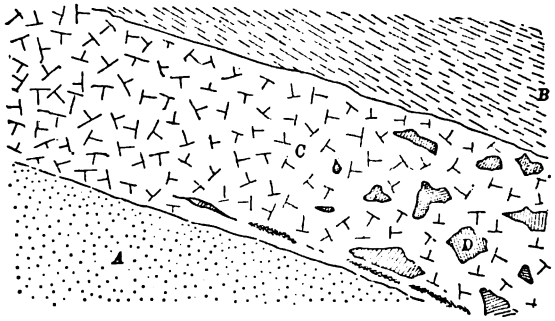


Fig. 10.



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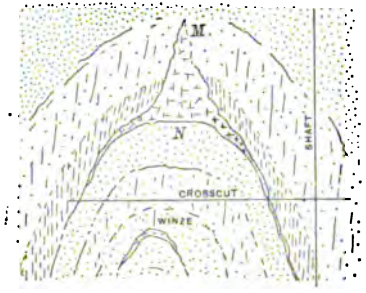

SHENANDOAH MINE.

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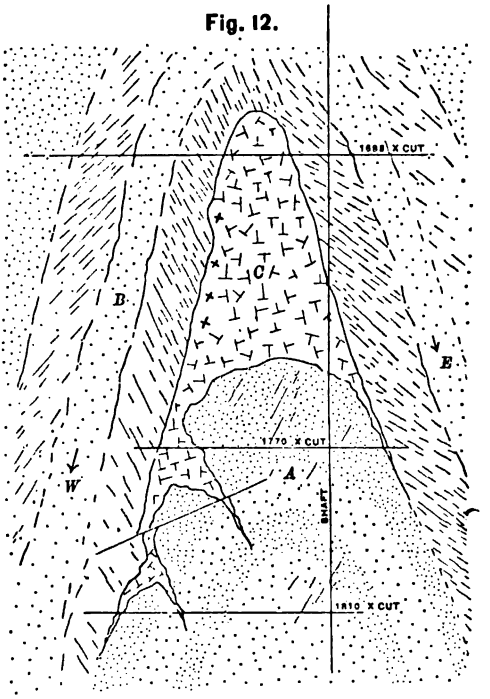

NEW CHUM CONSOLIDATED MINE.

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Shaft.	Line of Reef.	Ave. Strike.	Average pitch of formation.	Depth.	Distance of center-country from Shaft.	Depth.	Distance of center-country from Shaft.	Average dip.
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Victory and Pandora	Garden Gully	25° W. of N.	1 in 6 (at 650 ft.)	140	9 " "	2160	130 " "	1 in 15½
Gt. Ex. Hustlers.....	Hustlers.....	35° W of N.	1 in 5 (at 1800 ft.)	200	80 " "	1800	160 " "	1 in 6½

At the "180" mine, for instance, the strike of the New Chum formation is 21° W. of N., the pitch of the saddle at the 2500-foot level is northward 1 in 6, the dip of center-country is east about 1 in 14. The last is determined thus: at the 560-foot cross-cut the center country is 65 feet *west* of the shaft, while at the 2500 it is 70 feet *east* of the shaft, so that it has travelled eastward 135 feet in about 1940 feet, the general dip of the country having taken from one side to the other of this particular shaft the series of beds intersected by it. The ideal section, Fig. 4 (p. 471), represents the structure, and shows how, the general trend of the country being to the east, any one of the deep mine-workings will intercept, in succession, many saddles, some of which will prove gold-bearing and some barren. It will be understood that only the gold-bearing ones of the series

are developed in a manner permitting their proper examination. One mine may have only two gold-bearing saddles, while another may have exposed a dozen or more.* As soon as one formation has been worked out, when the legs intercepted by the deeper cross-cuts are found to become too small or too poor for profit, prospecting is renewed until another gold-bearing saddle is cut. Its apex may occur between the lower portions of the legs of the last saddle or it may not be found for 200 or 300 feet deeper. *Deeper*, not further east or west on the same horizon; for the working of the mine has

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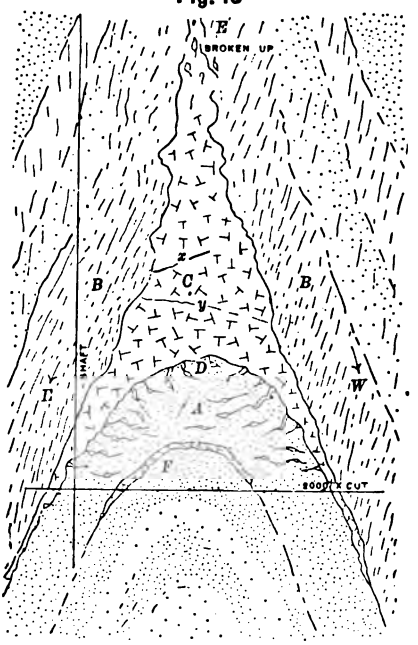


shown that a gold-bearing succession of saddles will go down in step-like gradations, each succeeding one being slightly eastward of the one above it (see Fig. 5, p. 473). This simplicity of arrangement is, of course, much complicated, sometimes utterly destroyed, by faulting, and by the formation of irregular bodies of quartz that may be recognized occasionally as imperfect saddles.

* In the New Chum and Victoria mine, for instance, as many as 30 saddles have been passed through from the surface to 2300 feet. In the "180" mine, five have been discovered and explored, of which number three have proved profitably auriferous.

average total dividends of each of these 20 properties* amounts to £81,947, the average nominal capital of each is £49,742, while the average paid up capital is only £31,167. It is seen, therefore, that these 20 companies have returned in dividends nearly twice the amount of capital called up for their equipment and development. Further, it should be stated that the number under consideration does not include several of the mines† which have the

* The following list gives the detailed figures of the twenty companies which were dividend-paying at the beginning of this year (1891). The results are given up to the end of 1890:

Company.	Nominal capital. £.	Paid up capital. £.	Dividends. £.
Catherine Reef United,	135,200	82,810	50,755
Fortuna Hustlers,	14,000	3,500	2,800
Gt. Extended Hustlers,	68,000	62,050	419,200
Hercules and Energetic,	60,000	8,250	74,625
Johnson's Reef,	78,000	72,100	223,950
Lazarus New Chum,	67,500	63,187	67,500
Lazarus No. 1,	67,500	63,950	95,062
Lady Barkly,	24,000	17,400	58,315
New Chum Con,	42,000	18,200	132,300
New Chum Railway,	36,890	28,589	52,078
New Chum United,	14,750	8,850	66,375
North Old Chum,	54,000	33,075	86,495
Rose of Denmark,	24,000	4,800	63,600
Shamrock,	45,000	27,750	15,750
Shenandoah,	96,000	33,600	59,600
South New Chum,	32,000	21,735	800
Specimen Hill United,	20,000	12,000	23,500
United Hustlers and Redan,	48,000	19,200	114,000
United Devonshire,	28,000	9,800	21,700
Young Chum,	40,000	32,500	10,550
Totals,	994,840	623,346	1,638,955
Averages,	49,742	31,167	81,947

† The greatest producers have been the following mines:

Name.	Called up capital. £.	Dividends. £.
Garden Gully United,	21,642	667,796
Gt. Extended Hustlers,	62,050	419,200
Johnson's Reef,	72,100	223,950
Gt. Hustler's Tribute,	61,200	620,200
North Johnson's,	31,850	148,625
United Devonshire,	8,244	224,000
New Chum Con.,	18,200	132,300

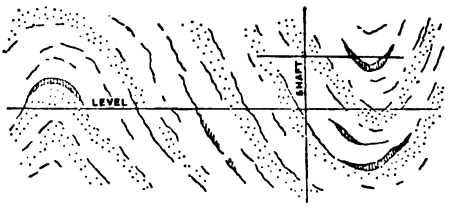
There are many others, the totals of whose dividends is more than £100,000.

greatest records, but have temporarily dropped out of the dividend-list.

The instances of individual productiveness here have been surpassed in other parts of the world; but it is doubtful whether in the history of gold-mining there can be shown a better record in the proportion of dividends paid to capital expended. The product of the mines from tribute-parties is not included in the above list unless especially mentioned. It is a matter of record, for instance, that, including the product of the tributers, the Garden Gully United has yielded over £1,000,000. The Kentish mine of the late J. B. Watson is said to have yielded over £2,500,000; and another private mine, G. Lansell's "180," has given magnificent profits to its owner, as have also several tribute-workings, the records of which are not accessible now.

To the Australian the name of Sandhurst is always associated with

Fig. 6.



INVERTED SADDLES.

the "saddle-reefs." It was the frequent mention of these which led me to visit the mines; and it is the peculiar interest which they must have for all geologists and mining engineers which has induced the publication of these notes. Before discussing, however, these and other matters of detail, a general account of the geology of the district should be given.

Geology.

As will be seen by reference to the map* of Victoria (Fig. 1, p. 465), the gold-field is situated near the northern edges of the exposure of the Lower Silurian, which, a short distance further north, is overlain by Pliocene shales. The boss of granite at Mt. Hope suggests

* The geological map of a portion of Victoria which accompanies this paper, was copied by me from the geological sketch-map in *The Geology and Physical Geography of Victoria*, by R. A. F. Murray, the geologist of the Victorian government.

the deeper masses of crystalline rocks which also form the Mt. Alexander ranges, dividing the Sandhurst and Castlemaine gold-fields. The gold-mining districts of Victoria are almost entirely confined to beds of Upper and Lower Silurian age, of which Mr. Murray says:* "As surface or underlying rocks, they occupy the greater part of Victoria from the sea-coast to elevations exceeding 6000 feet." They form the bed-rock of the alluvium the yield of which astonished the world in 1851, and the country-rock of the quartz-lodes from which that alluvial gold was derived. Selwyn† computed their total thickness to be not less than 35,000 feet. While the line of division between the two horizons has not been found, certain differences in lithological character and fossil remains have led the Victorian Geological Survey to refer to the Upper Silurian that portion of the Silurian rocks lying east of a line drawn from Melbourne to Heathcote, while the Silurian west of that line is regarded as Lower. It is in the lower horizon, which, according to Professor McCoy, corresponds to the Llandeilo Flags and Bala rocks of Wales, that the auriferous deposits of Ballarat, Castlemaine, and Sandhurst‡ occur. In the upper are the Ovens district, the Buckland and Harrietville. Again, to the latter belong the picturesque alpine districts, and to the former, the extensive plains diversified by low rounded hills.

While there is a general similiarity in the mode of occurrence of the gold, there are also some interesting differences in the habits of the quartz-lodes in the two horizons. The auriferous deposits are found traversing certain defined belts, which have a general strike 20° to 30° west of north. These parallel belts contain veins of quartz which conform to the general strike, and which, like the belts themselves, are separated by barren portions of country. My observations lead me to believe that the gold-veins of the Lower Silurian are more often conformable to the stratification, while those of the Upper are more frequently true fissures, traversing the country at an angle to the bedding. R. A. F. Murray, the government geologist, has noted that while the quartz-reefs in the Upper Silurian

* Page 33 of the book mentioned in the preceding note.

† A. R. C. Selwyn, formerly head of the Geological Survey of Victoria, and now occupying a similar position under the Canadian government.

‡ The fossils most common in the rocks at Sandhurst, are graptolites, particularly *stellatus*, *extensus* and *tripedes*, together with *Sertularia magna* and *vergata*, *Didymograpsus fruticosus*, and *Phyllograpsus folium*. In the weathered slate-beds they are easily found, but underground the dark color of the rock, and the development of fine cleavage renders them difficult of recognition.

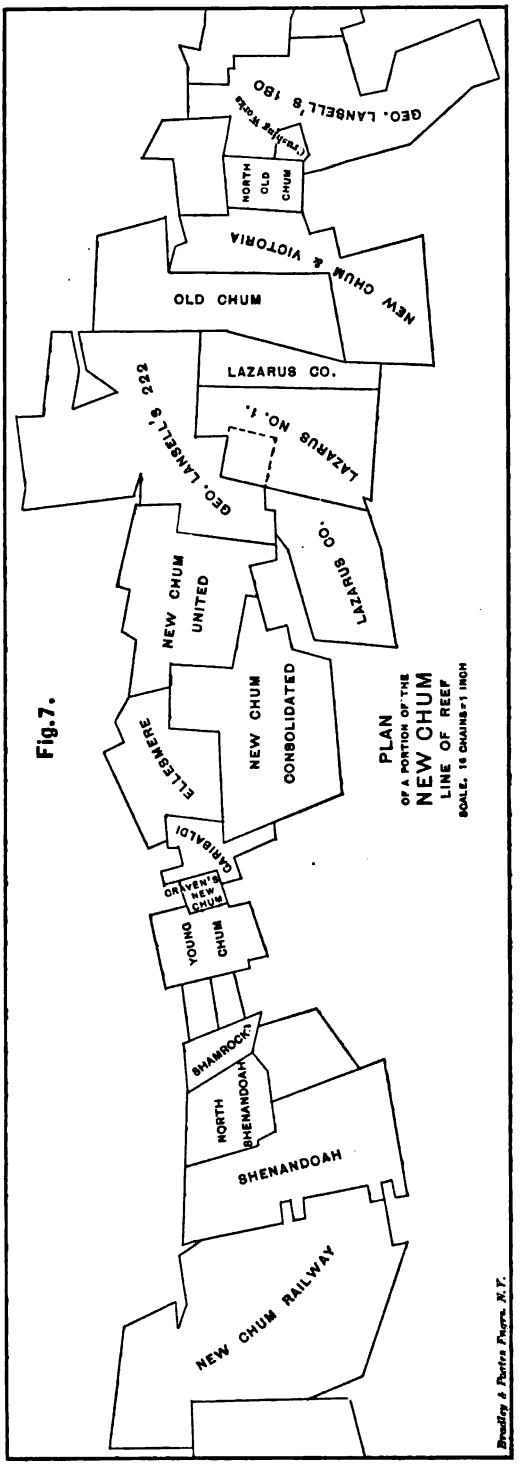


Fig. 7.

PLAN
OF A PORTION OF THE
NEW CHUM
LINE OF REEF
SCALE, 16 CHAINS=1 INCH

may be fewer in number and smaller in size, they have, on the other hand, considerably exceeded hitherto, in their average yield of gold, the reefs in the lower division.* It seems probable, however, that the smaller number of discovered reefs in the upper horizon is largely due to the more mountainous and less accessible character of the country in which they occur. In the few districts, such as Bright and the Buckland, where there has been any considerable amount of prospecting in the rocks of this division, large numbers of parallel veins have been exposed. Their greater yield per ton is accounted for by the fact that the ores of the mines in the Upper Silurian are frequently so charged with sulphides as to come under the "refractory" class, necessitating the working of ore of higher tenor than is required in the Lower Silurian, where the quartz is usually of the simplest "free-milling" type. This difference will also serve to explain the circumstance, mentioned by the government geologist, that all the largest nuggets obtained in the alluvial mines of Victoria have been found where Lower Silurian rocks prevail.

The Bendigo district consists, broadly speaking, of a belt of sandstones and slates of Lower Silurian age, abutting to the south and west against the granite of the Mt. Alexander ranges and overlain to the north and northeast by the shales of the Pliocene. The most marked characteristic of these sandstone and slate beds is the extreme bending, folding, and contortion which they have undergone, accompanied, as might be expected, by a varying amount of fissuring and faulting. The anticlinal and synclinal undulations are often remarkably sharp and exhibit every gradation in extent, from a few feet to miles, from a hand-specimen to those larger corrugations of the earth's crust which geologists name "ge-synclines" and "ge-anticlines." The main anticlinal axes strike N. N. W. and S. S. E., but there are also transverse undulations which further complicate the geological structure.

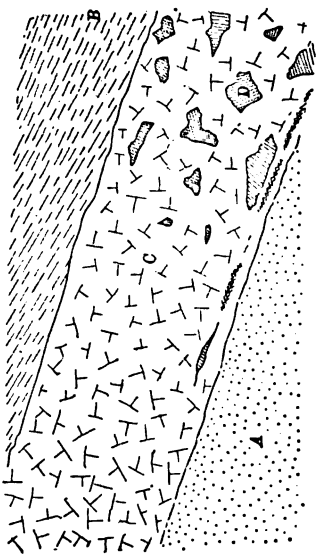
The Reefs.

The quartz reefs or lodes† conform to these folds in the country-

* The returns show that the average yield per ton is as follows:
Upper Silurian, Beechworth, 13 dwts. 23 grs., Gippsland 15 dwts. 21 grs.
Lower Silurian, Ballarat, 7 dwts. 21 grs., Bendigo, 9 dwts. 5 grs.

† The Australian calls a quartz lode or vein a "reef," the Californian a "ledge," while in Colorado the word "crevice" often does similar duty. Throughout this article I shall often use the colonial mining terms, since they are usually expressive and it would be difficult to find other names that are not also local.

Fig. 10.



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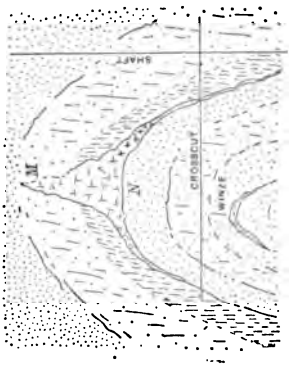

SHENANDOAH MINE.

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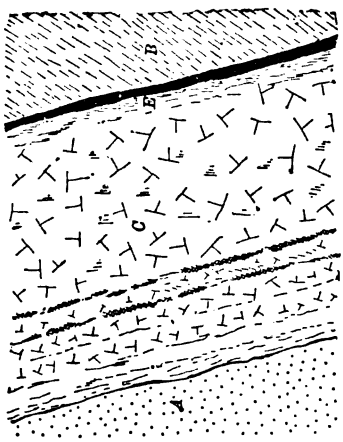

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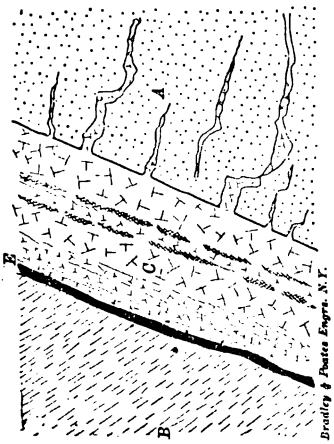

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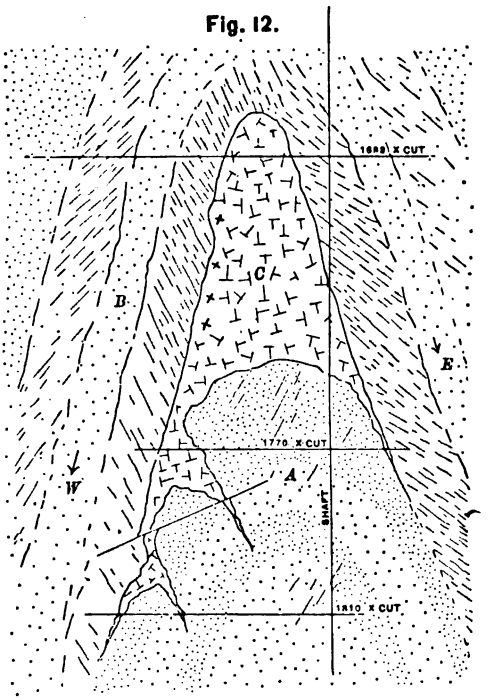

NEW CHUM CONSOLIDATED MINE.

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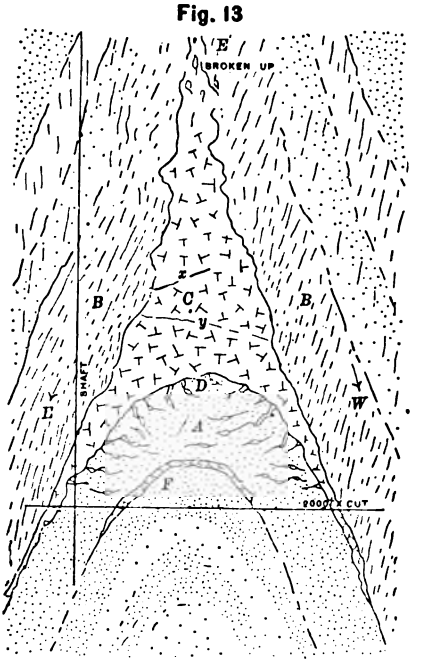

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average total dividends of each of these 20 properties* amounts to £81,947, the average nominal capital of each is £49,742, while the average paid up capital is only £31,167. It is seen, therefore, that these 20 companies have returned in dividends nearly twice the amount of capital called up for their equipment and development. Further, it should be stated that the number under consideration does not include several of the mines† which have the

* The following list gives the detailed figures of the twenty companies which were dividend-paying at the beginning of this year (1891). The results are given up to the end of 1890:

Company.	Nominal capital. £.	Paid up capital. £.	Dividends. £.
Catherine Reef United,	135,200	82,810	50,755
Fortuna Hustlers,	14,000	3,500	2,800
Gt. Extended Hustlers,	68,000	62,050	419,200
Hercules and Energetic,	60,000	8,250	74,625
Johnson's Reef,	78,000	72,100	223,950
Lazarus New Chum,	67,500	63,187	67,500
Lazarus No. 1,	67,500	63,950	95,062
Lady Barkly,	24,000	17,400	58,315
New Chum Con,	42,000	18,200	132,300
New Chum Railway,	36,890	28,589	52,078
New Chum United,	14,750	8,850	66,375
North Old Chum,	54,000	33,075	86,495
Rose of Denmark,	24,000	4,800	63,600
Shamrock,	45,000	27,750	15,750
Shenandoah,	96,000	33,600	59,600
South New Chum,	32,000	21,735	800
Specimen Hill United,	20,000	12,000	23,500
United Hustlers and Redan,	48,000	19,200	114,000
United Devonshire,	28,000	9,800	21,700
Young Chum,	40,000	32,500	10,550
Totals,	994,840	623,346	1,638,955
Averages,	49,742	31,167	81,947

† The greatest producers have been the following mines:

Name.	Called up capital. £.	Dividends. £.
Garden Gully United,	21,642	667,796
Gt. Extended Hustlers,	62,050	419,200
Johnson's Reef,	72,100	223,950
Gt. Hustler's Tribute,	61,200	620,200
North Johnson's,	31,850	148,625
United Devonshire,	8,244	224,000
New Chum Con.,	18,200	132,300

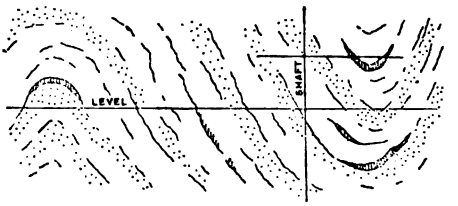
There are many others, the totals of whose dividends is more than £100,000.

greatest records, but have temporarily dropped out of the dividend-list.

The instances of individual productiveness here have been surpassed in other parts of the world; but it is doubtful whether in the history of gold-mining there can be shown a better record in the proportion of dividends paid to capital expended. The product of the mines from tribute-parties is not included in the above list unless especially mentioned. It is a matter of record, for instance, that, including the product of the tributers, the Garden Gully United has yielded over £1,000,000. The Kentish mine of the late J. B. Watson is said to have yielded over £2,500,000; and another private mine, G. Lansell's "180," has given magnificent profits to its owner, as have also several tribute-workings, the records of which are not accessible now.

To the Australian the name of Sandhurst is always associated with

Fig. 6.



INVERTED SADDLES.

the "saddle-reefs." It was the frequent mention of these which led me to visit the mines; and it is the peculiar interest which they must have for all geologists and mining engineers which has induced the publication of these notes. Before discussing, however, these and other matters of detail, a general account of the geology of the district should be given.

Geology.

As will be seen by reference to the map* of Victoria (Fig. 1, p. 465), the gold-field is situated near the northern edges of the exposure of the Lower Silurian, which, a short distance further north, is overlain by Pliocene shales. The boss of granite at Mt. Hope suggests

* The geological map of a portion of Victoria which accompanies this paper, was copied by me from the geological sketch-map in *The Geology and Physical Geography of Victoria*, by R. A. F. Murray, the geologist of the Victorian government.

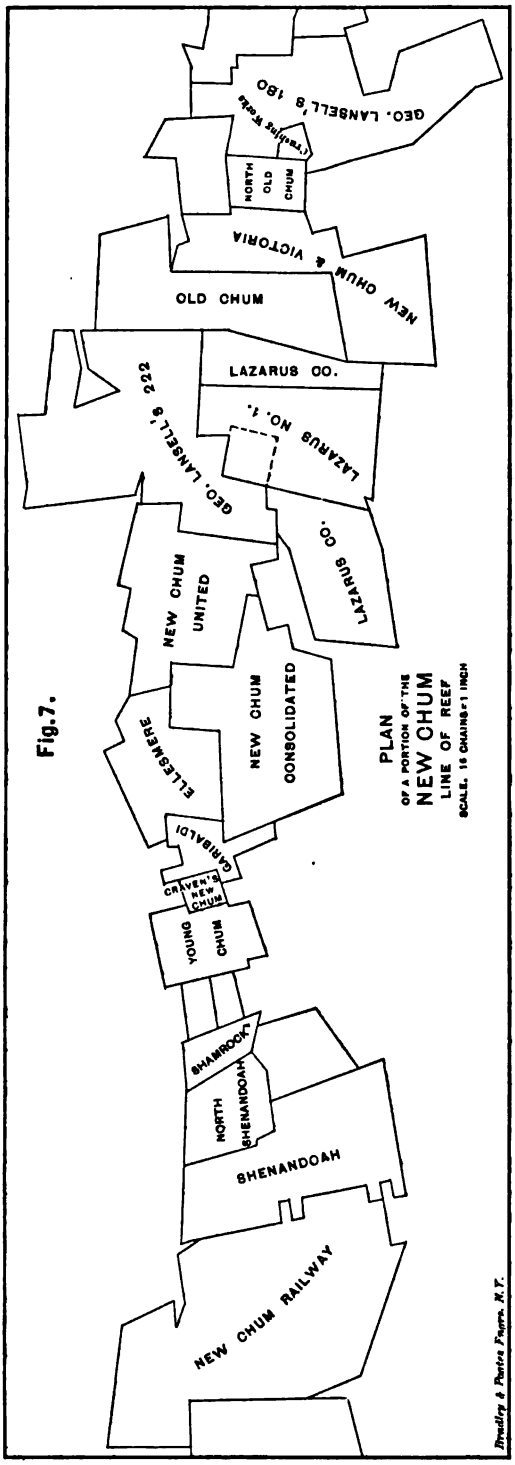
the deeper masses of crystalline rocks which also form the Mt. Alexander ranges, dividing the Sandhurst and Castlemaine gold-fields. The gold-mining districts of Victoria are almost entirely confined to beds of Upper and Lower Silurian age, of which Mr. Murray says:* "As surface or underlying rocks, they occupy the greater part of Victoria from the sea-coast to elevations exceeding 6000 feet." They form the bed-rock of the alluvium the yield of which astonished the world in 1851, and the country-rock of the quartz-lodes from which that alluvial gold was derived. Selwyn† computed their total thickness to be not less than 35,000 feet. While the line of division between the two horizons has not been found, certain differences in lithological character and fossil remains have led the Victorian Geological Survey to refer to the Upper Silurian that portion of the Silurian rocks lying east of a line drawn from Melbourne to Heathcote, while the Silurian west of that line is regarded as Lower. It is in the lower horizon, which, according to Professor McCoy, corresponds to the Llandeilo Flags and Bala rocks of Wales, that the auriferous deposits of Ballarat, Castlemaine, and Sandhurst‡ occur. In the upper are the Ovens district, the Buckland and Harrietville. Again, to the latter belong the picturesque alpine districts, and to the former, the extensive plains diversified by low rounded hills.

While there is a general similiarity in the mode of occurrence of the gold, there are also some interesting differences in the habits of the quartz-lodes in the two horizons. The auriferous deposits are found traversing certain defined belts, which have a general strike 20° to 30° west of north. These parallel belts contain veins of quartz which conform to the general strike, and which, like the belts themselves, are separated by barren portions of country. My observations lead me to believe that the gold-veins of the Lower Silurian are more often conformable to the stratification, while those of the Upper are more frequently true fissures, traversing the country at an angle to the bedding. R. A. F. Murray, the government geologist, has noted that while the quartz-reefs in the Upper Silurian

* Page 33 of the book mentioned in the preceding note.

† A. R. C. Selwyn, formerly head of the Geological Survey of Victoria, and now occupying a similar position under the Canadian government.

‡ The fossils most common in the rocks at Sandhurst, are graptolites, particularly *stellatus*, *extensus* and *tripedes*, together with *Sertularia magna* and *vergata*, *Didymograpsus fruticosus*, and *Phyllograpsus folium*. In the weathered slate-beds they are easily found, but underground the dark color of the rock, and the development of fine cleavage renders them difficult of recognition.


GEO. LANSELL'S 180
Freehold Reefs
NORTH OLD CHUM
NEW CHUM & VICTORIA
OLD CHUM
LAZARUS CO.
LAZARUS NO. 1.
GEO. LANSELL'S 222
NEW CHUM UNITED
LAZARUS CO.
ELLESMERE
NEW CHUM CONSOLIDATED
GARIBALDI
CRAVEN'S NEW CHUM
YOUNG CHUM
SHAMROCK
NORTH SHENANDOAH
SHENANDOAH
NEW CHUM RAILWAY
Fig.7.
PLAN
OF A PORTION OF THE
NEW CHUM
LINE OF REEF
SCALE, 15 CHAINS = 1 INCH
Bradley & Porter Engrs. N.Y.

may be fewer in number and smaller in size, they have, on the other hand, considerably exceeded hitherto, in their average yield of gold, the reefs in the lower division.* It seems probable, however, that the smaller number of discovered reefs in the upper horizon is largely due to the more mountainous and less accessible character of the country in which they occur. In the few districts, such as Bright and the Buckland, where there has been any considerable amount of prospecting in the rocks of this division, large numbers of parallel veins have been exposed. Their greater yield per ton is accounted for by the fact that the ores of the mines in the Upper Silurian are frequently so charged with sulphides as to come under the "refractory" class, necessitating the working of ore of higher tenor than is required in the Lower Silurian, where the quartz is usually of the simplest "free-milling" type. This difference will also serve to explain the circumstance, mentioned by the government geologist, that all the largest nuggets obtained in the alluvial mines of Victoria have been found where Lower Silurian rocks prevail.

The Bendigo district consists, broadly speaking, of a belt of sandstones and slates of Lower Silurian age, abutting to the south and west against the granite of the Mt. Alexander ranges and overlain to the north and northeast by the shales of the Pliocene. The most marked characteristic of these sandstone and slate beds is the extreme bending, folding, and contortion which they have undergone, accompanied, as might be expected, by a varying amount of fissuring and faulting. The anticlinal and synclinal undulations are often remarkably sharp and exhibit every gradation in extent, from a few feet to miles, from a hand-specimen to those larger corrugations of the earth's crust which geologists name "ge-synclines" and "ge-anticlines." The main anticlinal axes strike N. N. W. and S. S. E., but there are also transverse undulations which further complicate the geological structure.

The Reefs.

The quartz reefs or lodes† conform to these folds in the country-

* The returns show that the average yield per ton is as follows:
Upper Silurian, Beechworth, 13 dwts. 23 grs., Gippsland 15 dwts. 21 grs.
Lower Silurian, Ballarat, 7 dwts. 21 grs., Bendigo, 9 dwts. 5 grs.

† The Australian calls a quartz lode or vein a "reef," the Californian a "ledge," while in Colorado the word "crevice" often does similar duty. Throughout this article I shall often use the colonial mining terms, since they are usually expressive and it would be difficult to find other names that are not also local.

Fig. 8.

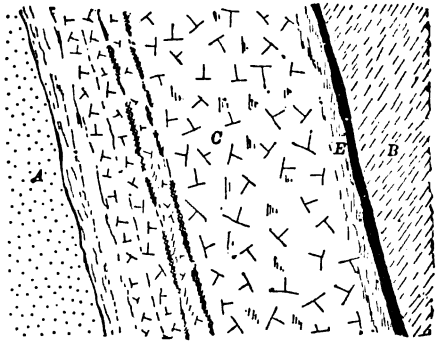
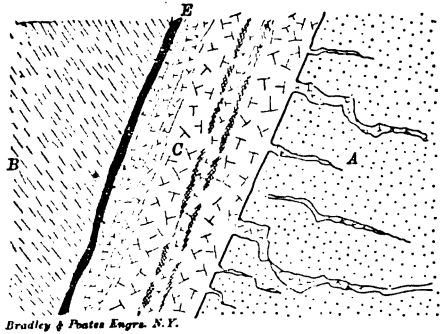


Fig. 9.



SHENANDOAH MINE.

Fig. 10.

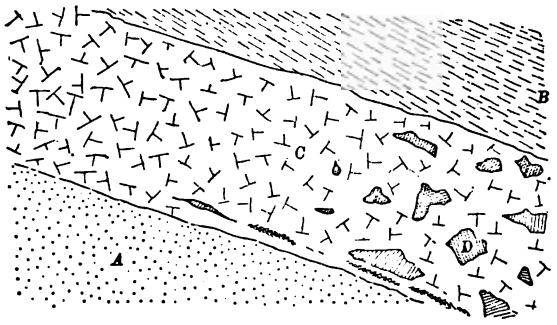
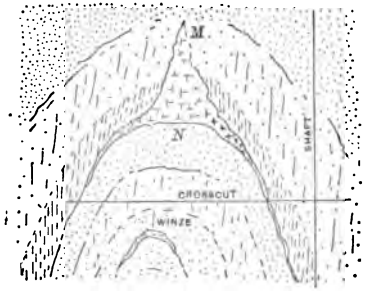


Fig. 11.



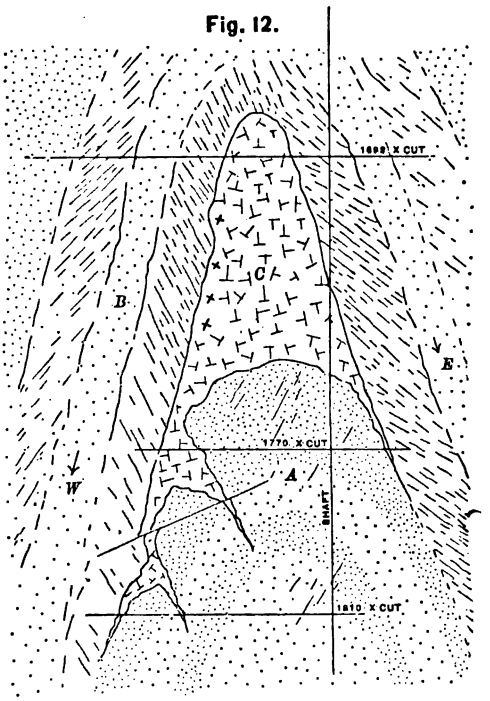
rock, that is, the ore-deposits lie between and along the beds of slate and sandstone, the anticlinal axes of which form the apex or cap of the quartz-formations, which are thus known as "saddles," while the lower portions, called the "legs," similarly dip east and west with the inclosing strata. In like manner, in the direction of their strike, the quartz-formations* pitch north and south conformably to the longitudinal undulations produced in the anticlinal axes by the transverse folds mentioned above.

The "saddle-reef" is the distinctive ore-deposit of the Sandhurst mines. The references to this most beautiful type of ore-deposit to be found among the works relating to the distribution and extraction of the precious metals, are both meager and inaccurate. In the Colonies, it is true, one hears a great deal of these "Sandhurst saddles," such references being, however, for the most part very vague and incorrect. In other mining districts, in the neighboring colonies of New South Wales, Queensland, etc., the writer has often heard this or that mine spoken of as containing "a saddle, just like those of Sandhurst." On examination these proved in every case to be different forms of the ordinary junction of two lodes, not true anticlines, producing however bodies of quartz which the old Bendigo digger would promptly label as "saddles," for the sake, perhaps, of *auld lang syne*. I was beginning to fear that the saddle-reef as a distinct formation did not exist at all, even at Sandhurst, until underground, at the New Chum and Victoria mine, I saw for the first time a type of ore-deposit which is perhaps the most interesting of the many forms in which gold is known to occur. I remember well my delight in recognizing the peculiar character of the lodes, also my disappointment on finding that the "west legs" did not conform to the bedding, and then finally my relief when I saw that in the latter observation I had confounded bedding and cleavage. The cleavage in some parts of the field is so strong as to obliterate the bedding; and it is by reason of this fact that so many observers have gone astray. In what is practically the only authority dealing with Victorian mining—Brough Smyth's *Gold-Fields of Victoria*—there is a lamentable confusion upon this point. The difficulty is further increased by the fact that there are "false saddles," one leg

* In the mines, the words "pitch," "dip," "underlay," etc., are used indiscriminately. I shall always use "dip" to express the angle with the horizon, east or west, made by the beds of the country or the reefs which are conformable to them. "Pitch" will be used to describe the inclination north or south along the strike of the quartz-formations.

of which conforms to the stratification, while the other follows a joint or some other cross-fracture in the country. Figs. 2 and 3 (pp. 467 and 469), illustrate the simplest type of true and false saddles. In both cases the bedding-planes are indicated, while, to avoid confusion, the lines of cleavage are omitted. In Fig. 2, A would be called the cap or apex, B the west leg, and C the east leg. A cross-cut passing through D would be said to be in "center-country;" as soon as the dip of the beds became distinctly east or west the cross-

Fig. 12.



NEW CHUM CONSOLIDATED MINE.

cut would have penetrated into "east" or "west country." In Fig. 3, A A illustrates a fissure, sometimes a fault, sometimes only a joint; D D is the ore formed along this line of fissure, while B is the body of quartz formed at its junction with the bedding-plane C C, which carries another vein of quartz. When a formation like this is further complicated by a few minor faults and the development of a strong slaty cleavage at varying angles, it is very difficult to determine correctly the true facts of the case; and the quartz-body may be mistaken for a true saddle, that is, an anticline of quartz.

The mines are located along the various "lines of reef," which are coincident with the surface-exposure of the quartz formed along the anticlinal axes. The general dip of the country (as distinguished from that of the individual beds comprising it) is eastward; or, as the miners put it, "center-country dips east." The following statement illustrates this point. Three mines (they happen to be three famous producers) are taken approximately opposite each other on the three great lines of reef—the New Chum, the Garden Gully, and the Hustlers. In the accompanying table, the third column gives the average strike of the anticline which forms the "line of reef," and the fourth the pitch of the quartz-formation at the depths indicated in the brackets. The last column gives the dip of "center-country" as determined from the figures given in columns 5 to 8. Taking the saddle as similar to the roof of a house, the third column gives the angle which the ridge makes with the meridian; the fourth, the angle with the plane of the horizon; and the last column, the inclination of its axis when it is slightly tilted to one side.

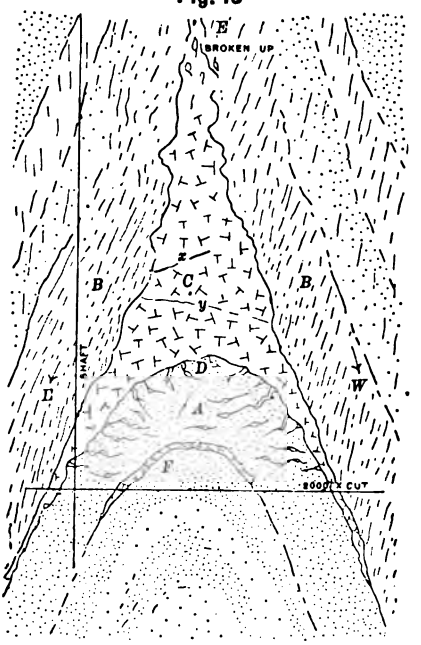
Table showing the Principal Features of the Main Saddle-Formations.

Shaft.	Line of Reef.	Ave. Strike.	Average pitch of formation.	Depth.	Distance of center-country from Shaft.	Depth.	Distance of center-country from Shaft.	Average dip.
"180" Mine.....	New Chum...	21° W. of N.	1 in 6 (at 2500 ft.)	560	65 ft. W.	2500	70 ft. E.	1 in 14
Victory and Pandora	Garden Gully	25° W. of N.	1 in 6 (at 650 ft.)	140	9 " "	2160	130 " "	1 in 15½
Gt. Ex. Hustlers.....	Hustlers.....	35° W of N.	1 in 5 (at 1800 ft.)	200	80 " "	1800	160 " "	1 in 6½

At the "180" mine, for instance, the strike of the New Chum formation is 21° W. of N., the pitch of the saddle at the 2500-foot level is northward 1 in 6, the dip of center-country is east about 1 in 14. The last is determined thus: at the 560-foot cross-cut the center country is 65 feet *west* of the shaft, while at the 2500 it is 70 feet *east* of the shaft, so that it has travelled eastward 135 feet in about 1940 feet, the general dip of the country having taken from one side to the other of this particular shaft the series of beds intersected by it. The ideal section, Fig. 4 (p. 471), represents the structure, and shows how, the general trend of the country being to the east, any one of the deep mine-workings will intercept, in succession, many saddles, some of which will prove gold-bearing and some barren. It will be understood that only the gold-bearing ones of the series

are developed in a manner permitting their proper examination. One mine may have only two gold-bearing saddles, while another may have exposed a dozen or more.* As soon as one formation has been worked out, when the legs intercepted by the deeper cross-cuts are found to become too small or too poor for profit, prospecting is renewed until another gold-bearing saddle is cut. Its apex may occur between the lower portions of the legs of the last saddle or it may not be found for 200 or 300 feet deeper. *Deeper*, not further east or west on the same horizon; for the working of the mine has

Fig. 13



222 AND LAZARUS MINES.

shown that a gold-bearing succession of saddles will go down in step-like gradations, each succeeding one being slightly eastward of the one above it (see Fig. 5, p. 473). This simplicity of arrangement is, of course, much complicated, sometimes utterly destroyed, by faulting, and by the formation of irregular bodies of quartz that may be recognized occasionally as imperfect saddles.

* In the New Chum and Victoria mine, for instance, as many as 30 saddles have been passed through from the surface to 2300 feet. In the "180" mine, five have been discovered and explored, of which number three have proved profitably auriferous.

As would be expected by the geologist, these anticlines alternate with synclinal undulations; but this fact has been recognized by few of those engaged in the development of the field, though it is a point of paramount importance to the proper conception of the mode of occurrence of the quartz. The explanation of this matter is that exploration has been confined, particularly in the deep mines on the New Chum reef, to a narrow strip of country in the immediate vicinity of the great anticlinal axes; and, though it is known that quartz lodes occur, for instance, between the New Chum and Garden Gully lines, these "side-lines," as they are called, have been neglected,* and it is in that portion of the country that the synclines would be situated. Where extensive faulting has taken place, these synclines are to be seen in the workings of some of the mines on the main "lines of reef." In the Confidence Extended, for instance, I observed some small "inverted saddles," as also in the Hercules and Energetic. In both cases there is reason to believe that a dislocation has brought a portion of the west country into line with one of the chief anticlinal axes, in the one case the Garden Gully, in the other the New Chum. A good instance of the more frequent occurrence of "inverted saddles" is presented by the Great Britain mine, located on the Carshalton reef, a "side-line" west of the New Chum. Fig. 6 (p. 475) illustrates it.

The distribution of the gold in the quartz of the saddle-reefs shows great variations. In a given formation in one mine the cap or a portion of the cap only will pay, while on the same formation in a neighboring claim the cap and west leg may yield good returns, or again both legs may prove gold-bearing while the cap is barren. The quartz-bodies do not extend uninterruptedly along their strike; their continuity is broken by overlappings or disturbed by faults. Different local changes in the structure of the country give very different shapes to the enclosed formations.

The most extensive ore-body and the greatest yield of gold from any one formation is claimed by the Garden Gully line from a saddle which traversed a group of mines (of which the Garden Gully United and the Victory and Pandora were the chief producers), at

* Since writing the above, and during a later visit to Sandhurst, I learn that the recent legislation as to mining on private property, together with the notable success of the New Red, White and Blue Consolidated, which is on a "side line" (the Sheepshead), has caused the beginning of active exploration on these subsidiary formations, this exploration being due, as has often been the case in this gold-field, to the enterprise of Mr. Lansell.

Fig. 15.

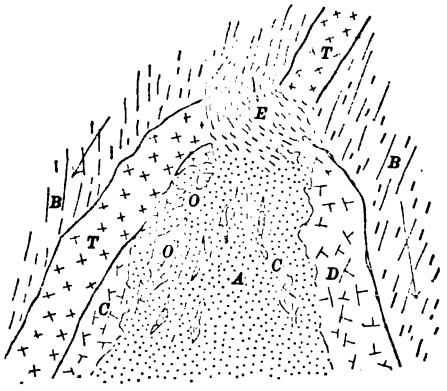


Fig. 16.

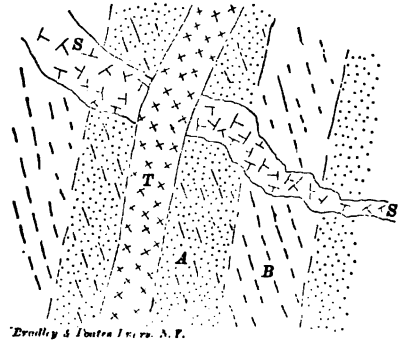


Fig. 14.

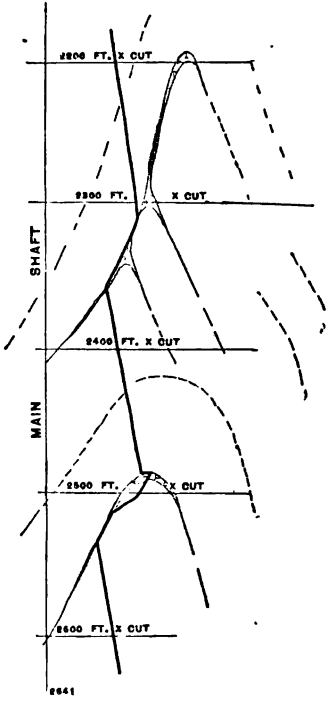


Fig. 17.

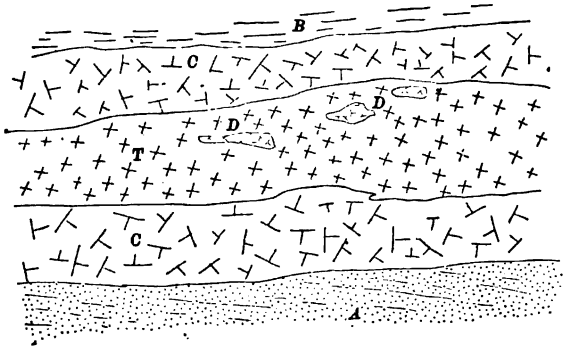
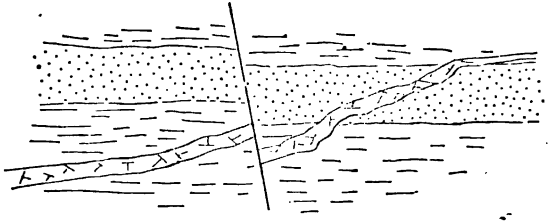


Fig. 18.



Vertical Section.
"180" Mine; Scale 100 feet $\frac{1}{4}$ inch.

a depth of from 600 to 700 feet from the surface. In the Garden Gully United alone, it was worked for 646 yards along the strike during 14 years, and yielded 13 tons of gold. For continuous productiveness, most numerous succession of gold-bearing formations, and greatest underground development, the New Chum ranks first. One can walk underground from the Victoria Consols to Golden Square, through the continuous workings of 21 mines, at a depth of from 1800 to 2000 feet, for a distance of 2 miles. The average number of "payable" saddles in any one mine on the New Chum line from the surface to 2000 feet, would be not far short of ten; on the Garden Gully the gold has been practically confined to two formations. The Hustlers line had its greatest development in the Great Extended Hustlers, the Hustlers, and the United Hustlers and Redan. It has been worked less extensively than the other two main lines of reef; but its known extent is great, and it should prove an important producer in years to come. Of the various formations to be seen in the mines of Sandhurst at the time of my inspection, the largest, most extensive, and most profitable was that on the New Chum lode between the New Chum railway and the "180" mine. It extended through 14 different claims (see Fig. 7, p. 477), a distance of 2400 yards, and was intercepted at depths which varied from 1600 to 2400 feet. I saw the development of this formation in six of the mines through which it passed, and, with the aid of drawings reproduced from sketches made underground, I will endeavor to describe the different modes of its occurrence.

The Shenandoah Mine.

The most southerly mine on this great body of gold-bearing quartz is the New Chum Railway, which, on account of the good returns obtained from this formation at 2025 feet depth, is often quoted in the Colonies as an example of deep gold-mining. From this company's ground the formation rises towards the Shenandoah, where at 1990 feet* the west leg was being very profitably exploited at the time of my visit. In the breast of the level and in the stopes overhead, the reef is seen to be notably laminated† and very well defined. As the cap is approached the walls remain as before, but the quartz becomes broken, dead-white in color, and with a splintery

* It will be seen that between the New Chum Railway and the Shenandoah, the back or anticlinal axis of the formation pitches about 100 feet in 600.

† In Amador county, Cal., this would be called "ribbon-rock." In both California and Australia such a structure is considered a favorable sign.

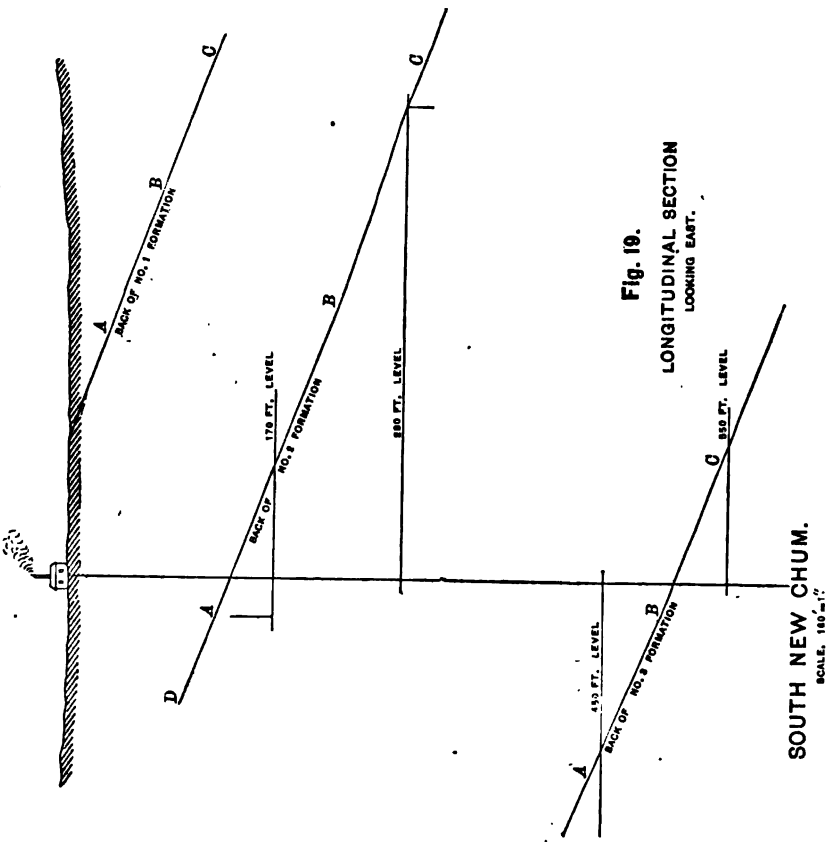


Fig. 19.
LONGITUDINAL SECTION
LOOKING EAST.

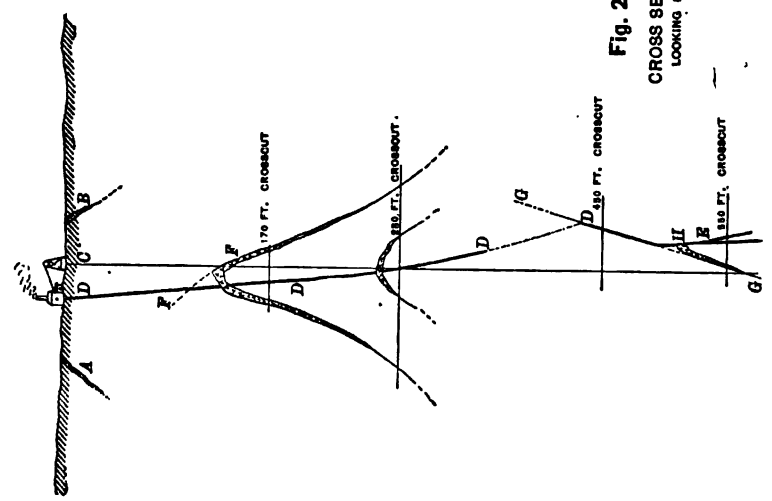


Fig. 20.
CROSS SECTION
LOOKING NORTH.

may be fewer in number and smaller in size, they have, on the other hand, considerably exceeded hitherto, in their average yield of gold, the reefs in the lower division.* It seems probable, however, that the smaller number of discovered reefs in the upper horizon is largely due to the more mountainous and less accessible character of the country in which they occur. In the few districts, such as Bright and the Buckland, where there has been any considerable amount of prospecting in the rocks of this division, large numbers of parallel veins have been exposed. Their greater yield per ton is accounted for by the fact that the ores of the mines in the Upper Silurian are frequently so charged with sulphides as to come under the "refractory" class, necessitating the working of ore of higher tenor than is required in the Lower Silurian, where the quartz is usually of the simplest "free-milling" type. This difference will also serve to explain the circumstance, mentioned by the government geologist, that all the largest nuggets obtained in the alluvial mines of Victoria have been found where Lower Silurian rocks prevail.

The Bendigo district consists, broadly speaking, of a belt of sandstones and slates of Lower Silurian age, abutting to the south and west against the granite of the Mt. Alexander ranges and overlain to the north and northeast by the shales of the Pliocene. The most marked characteristic of these sandstone and slate beds is the extreme bending, folding, and contortion which they have undergone, accompanied, as might be expected, by a varying amount of fissuring and faulting. The anticlinal and synclinal undulations are often remarkably sharp and exhibit every gradation in extent, from a few feet to miles, from a hand-specimen to those larger corrugations of the earth's crust which geologists name "ge-synclines" and "ge-anticlines." The main anticlinal axes strike N. N. W. and S. S. E., but there are also transverse undulations which further complicate the geological structure.

The Reefs.

The quartz reefs or lodes† conform to these folds in the country-

* The returns show that the average yield per ton is as follows:
Upper Silurian, Beechworth, 13 dwts. 23 grs., Gippsland 15 dwts. 21 grs.
Lower Silurian, Ballarat, 7 dwts. 21 grs., Bendigo, 9 dwts. 5 grs.

† The Australian calls a quartz lode or vein a "reef," the Californian a "ledge," while in Colorado the word "crevice" often does similar duty. Throughout this article I shall often use the colonial mining terms, since they are usually expressive and it would be difficult to find other names that are not also local.

Fig. 8.

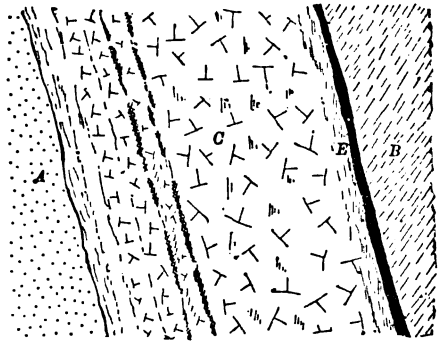
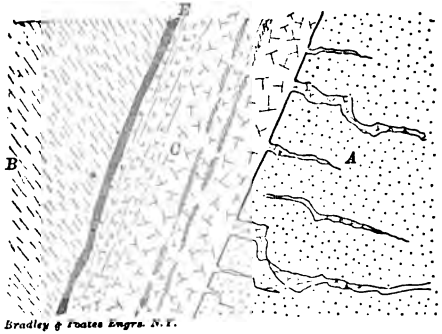


Fig. 9.



SHENANDOAH MINE.

Fig. 10.

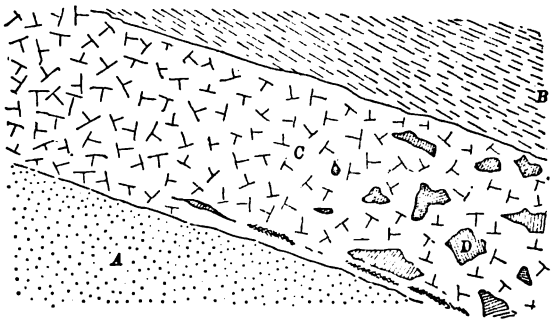
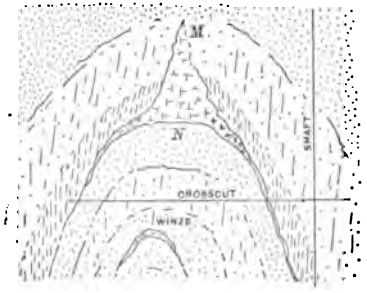


Fig. 11.



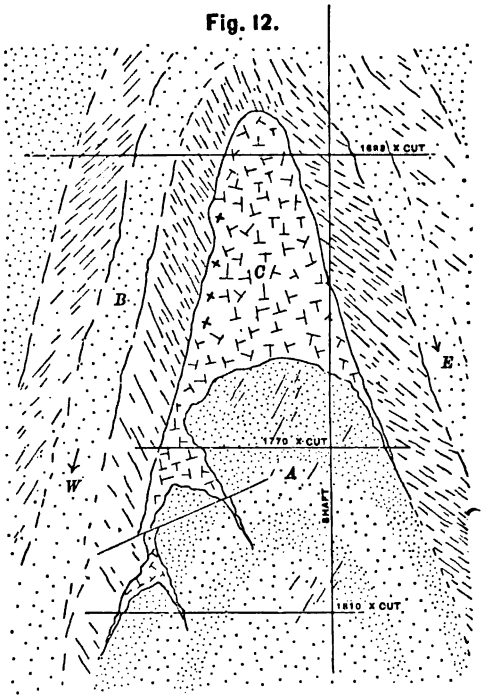
rock, that is, the ore-deposits lie between and along the beds of slate and sandstone, the anticlinal axes of which form the apex or cap of the quartz-formations, which are thus known as "saddles," while the lower portions, called the "legs," similarly dip east and west with the inclosing strata. In like manner, in the direction of their strike, the quartz-formations* pitch north and south conformably to the longitudinal undulations produced in the anticlinal axes by the transverse folds mentioned above.

The "saddle-reef" is the distinctive ore-deposit of the Sandhurst mines. The references to this most beautiful type of ore-deposit to be found among the works relating to the distribution and extraction of the precious metals, are both meager and inaccurate. In the Colonies, it is true, one hears a great deal of these "Sandhurst saddles," such references being, however, for the most part very vague and incorrect. In other mining districts, in the neighboring colonies of New South Wales, Queensland, etc., the writer has often heard this or that mine spoken of as containing "a saddle, just like those of Sandhurst." On examination these proved in every case to be different forms of the ordinary junction of two lodes, not true anticlines, producing however bodies of quartz which the old Bendigo digger would promptly label as "saddles," for the sake, perhaps, of *auld lang syne.* I was beginning to fear that the saddle-reef as a distinct formation did not exist at all, even at Sandhurst, until underground, at the New Chum and Victoria mine, I saw for the first time a type of ore-deposit which is perhaps the most interesting of the many forms in which gold is known to occur. I remember well my delight in recognizing the peculiar character of the lodes, also my disappointment on finding that the "west legs" did not conform to the bedding, and then finally my relief when I saw that in the latter observation I had confounded bedding and cleavage. The cleavage in some parts of the field is so strong as to obliterate the bedding; and it is by reason of this fact that so many observers have gone astray. In what is practically the only authority dealing with Victorian mining—Brough Smyth's *Gold-Fields of Victoria*—there is a lamentable confusion upon this point. The difficulty is further increased by the fact that there are "false saddles," one leg

* In the mines, the words "pitch," "dip," "underlay," etc., are used indiscriminately. I shall always use "dip" to express the angle with the horizon, east or west, made by the beds of the country or the reefs which are conformable to them. "Pitch" will be used to describe the inclination north or south along the strike of the quartz-formations.

of which conforms to the stratification, while the other follows a joint or some other cross-fracture in the country. Figs. 2 and 3 (pp. 467 and 469), illustrate the simplest type of true and false saddles. In both cases the bedding-planes are indicated, while, to avoid confusion, the lines of cleavage are omitted. In Fig. 2, A would be called the cap or apex, B the west leg, and C the east leg. A cross-cut passing through D would be said to be in "center-country;" as soon as the dip of the beds became distinctly east or west the cross-

Fig. 12.



NEW CHUM CONSOLIDATED MINE.

cut would have penetrated into "east" or "west country." In Fig. 3, A A illustrates a fissure, sometimes a fault, sometimes only a joint; D D is the ore formed along this line of fissure, while B is the body of quartz formed at its junction with the bedding-plane C C, which carries another vein of quartz. When a formation like this is further complicated by a few minor faults and the development of a strong slaty cleavage at varying angles, it is very difficult to determine correctly the true facts of the case; and the quartz-body may be mistaken for a true saddle, that is, an anticline of quartz.

The mines are located along the various "lines of reef," which are coincident with the surface-exposure of the quartz formed along the anticlinal axes. The general dip of the country (as distinguished from that of the individual beds comprising it) is eastward; or, as the miners put it, "center-country dips east." The following statement illustrates this point. Three mines (they happen to be three famous producers) are taken approximately opposite each other on the three great lines of reef—the New Chum, the Garden Gully, and the Hustlers. In the accompanying table, the third column gives the average strike of the anticline which forms the "line of reef," and the fourth the pitch of the quartz-formation at the depths indicated in the brackets. The last column gives the dip of "center-country" as determined from the figures given in columns 5 to 8. Taking the saddle as similar to the roof of a house, the third column gives the angle which the ridge makes with the meridian; the fourth, the angle with the plane of the horizon; and the last column, the inclination of its axis when it is slightly tilted to one side.

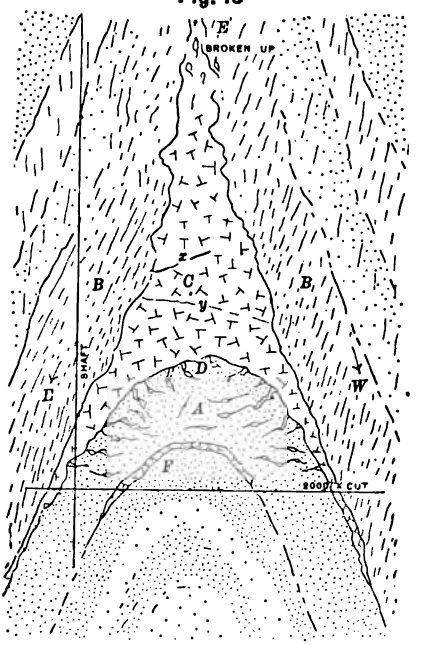
Table showing the Principal Features of the Main Saddle-Formations.

Shaft.	Line of Reef.	Ave. Strike.	Average pitch of formation.	Depth.	Distance of center-country from Shaft.	Depth.	Distance of center-country from Shaft.	Average dip.
"180" Mine.....	New Chum...	21° W. of N.	1 in 6 (at 2500 ft.)	560	65 ft. W.	2500	70 ft.E.	1 in 14
Victory and Pandora	Garden Gully	25° W. of N.	1 in 6 (at 650 ft.)	140	9 " "	2160	130 " "	1 in 15½
Gt. Ex. Hustlers.....	Hustlers.....	35° W of N.	1 in 5 (at 1800 ft.)	200	80 " "	1800	160 " "	1 in 6½

At the "180" mine, for instance, the strike of the New Chum formation is 21° W. of N., the pitch of the saddle at the 2500-foot level is northward 1 in 6, the dip of center-country is east about 1 in 14. The last is determined thus: at the 560-foot cross-cut the center country is 65 feet *west* of the shaft, while at the 2500 it is 70 feet *east* of the shaft, so that it has travelled eastward 135 feet in about 1940 feet, the general dip of the country having taken from one side to the other of this particular shaft the series of beds intersected by it. The ideal section, Fig. 4 (p. 471), represents the structure, and shows how, the general trend of the country being to the east, any one of the deep mine-workings will intercept, in succession, many saddles, some of which will prove gold-bearing and some barren. It will be understood that only the gold-bearing ones of the series

are developed in a manner permitting their proper examination. One mine may have only two gold-bearing saddles, while another may have exposed a dozen or more.* As soon as one formation has been worked out, when the legs intercepted by the deeper cross-cuts are found to become too small or too poor for profit, prospecting is renewed until another gold-bearing saddle is cut. Its apex may occur between the lower portions of the legs of the last saddle or it may not be found for 200 or 300 feet deeper. *Deeper*, not further east or west on the same horizon; for the working of the mine has

Fig. 13



222 AND LAZARUS MINES.

shown that a gold-bearing succession of saddles will go down in step-like gradations, each succeeding one being slightly eastward of the one above it (see Fig. 5, p. 473). This simplicity of arrangement is, of course, much complicated, sometimes utterly destroyed, by faulting, and by the formation of irregular bodies of quartz that may be recognized occasionally as imperfect saddles.

* In the New Chum and Victoria mine, for instance, as many as 30 saddles have been passed through from the surface to 2300 feet. In the "180" mine, five have been discovered and explored, of which number three have proved profitably auriferous.

The North Old Chum Mine.

Next to the Lazarus,* is the Old Chum, which I did not visit, and then follows the New Chum and Victoria, which was not deep enough† to cut the formation which we have been following through the different mines. Then follows the North Old Chum, where, at 2290 feet, the main formation has been profitably developed. A cross-cut at that depth cut the west leg at 70 feet from the shaft, and 45 feet further intercepted the east leg. A lava dike about 4 inches wide is seen 24 feet east of the west leg. The west reef has been driven upon for about 20 feet each way. In the north end it was 8 feet wide, with clean walls, carrying coarse‡ gold, easily visible in the quartz which had the same dead-white color and curious fracture as that of the Lazarus. There were to be seen patches of mineral (mostly arsenical pyrites) arranged along the black lines formed by some included slate. From this drift, during my stay in Sandhurst, was broken stone which gave "the deepest dividends" on record for the Southern hemisphere.§

In the south end the lode was smaller, as would be expected, since the saddle pitches north. The width was 2 feet in the face, and the back of the level showed coarse gold, the quartz being similar to that of the north drift and accompanied by 2 inches of black clay or "dig." The hanging-wall, as heretofore in the other mines, is seen to be slate, while immediately under the foot-wall there is "corduroy" or closely alternating thin beds of sandstone of varying hardness and composition. Near the lode the country shows coarse pyrites, arranged with no apparent regularity.

The east leg had not been driven upon when I saw it; but since then it has been developed and has given good stone.

When the north drift from the 2290-foot level in the North Old Chum has been driven 300 feet further, it will connect with the 2300-foot level in Lansell's "180" mine,‖ the deepest in Australia.

* I take this opportunity to thank the mine-manager, Mr. T. Whitford, for his courtesy in taking me through the workings.

† Since my visit the shaft has been sunk and the reef cut at 2260 feet.

‡ The coarse character of the gold is indicated by the yield of 223 oz. of gold from only 341 oz. of amalgam.

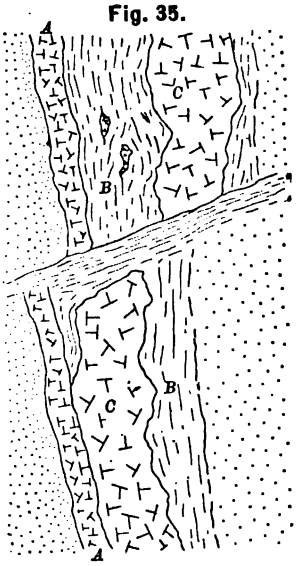
§ For the week ending September 13, 1890, it yielded 223 oz. 9 dwts. from 141 tons, which result enabled a sixpenny dividend to be paid. The depth was 2290 feet.

‖ The opportunity is offered me in this connection to express my great obligations to Mr. George Lansell, the man who has done most to develop the Sandhurst district, for assistance given me in visiting the mines, as well as for valuable information regarding mining and milling on the gold-field.

The "180" Mine.

In this mine the crown or cap of the saddle is cut by the 2200-foot cross-cut, 80 feet east from the shaft, showing a width of 12 feet of quartz. At 43 feet a dike is cut. In the 2300-foot cross-cut the west leg is cut at 58 feet, while the east leg is seen, broken up and irregular, at 130 feet. The dike is cut at 56 feet. In the 2400-foot cross-cut the west leg is 8 feet east of the shaft and the dike 46 feet. In the 2500-foot cross-cut this formation has not been cut, but the west leg of *another* saddle is found at 48 feet east, the corresponding east leg being at 81 feet. The dike is at 58 feet. In the 2600-foot cross-cut the same dike occurs at 56 feet. The cross-section (Fig. 14, p. 485) illustrates these formations. The dikes, one of which

Fig. 35.



Fault as Seen in the North End of the 1280-Foot Level of the Johnson's Mine.

is shown in the section, are a most important and interesting feature of the mining at Sandhurst, and will be referred to frequently.

In the "180" mine we see the furthest northern development of the great saddle which we have followed through several mines and which is first cut by the New Chum Railway. In the "180" mine it has not been opened up much; but at 2400 feet, in the south level, it forms the deepest gold-bearing quartz now worked in the Colonies. The west leg, the only portion of the formation which is worked at present, shows about 1 foot of very beautifully laminated

quartz. The quartz is brittle and flaky, the gold coarse and associated with a good percentage of iron pyrites and zinc-blende. On the hanging-wall there is a black graphitic slate which carries a noteworthy amount of mundic. The foot-wall shows cross-joints about 30 inches apart and nearly at right angles to the strike. The country is "corduroy," similar to that noted in the North Old Chum, and carries pyrites near the reef. The cleavage is highly developed, and to the east (about 60 feet) in the cross-cut, it obliterates the bedding, though the latter is to be distinguished with difficulty to the east near the east leg and to the west near the west leg. Between them the country is broken up and penetrated by the dike, which has a slight easterly dip.

The illustrations represent other interesting features of the "180" mine. Fig. 15 (p. 485) shows the point where the lava dike, about 8 inches wide, strikes the top of the saddle above the 2500-foot level. The arrows indicate the bedding; the slanting lines, the cleavage; E is disordered country. The dike T appears to be cut off, but it finds a passage in a plane other than that in which the section is taken. A is broken sandy rock. O and C are fragments of the cap. Fig. 16 (p. 485) is taken in center-country at the 2500-foot level. The dike T (here 10 inches wide) cuts through the spurs preparatory to taking a more regular course conformable to the bedding. In Fig. 14, already referred to, the dark line indicates the dike, which, though it rarely exceeds 9 inches in width, has been traced through the half-mile of workings from the surface down. It can be seen in the 2600-foot cross-cut and it is a noticeable feature of a surface cutting, a sketch of which is reproduced in Fig. 63 (p. 527). Fig. 17 (p. 485) shows the dike T where it forms a division in the quartz of the west leg (of the 2500-foot saddle) and carries included pieces of quartz, D. In this figure B is slate, A is slaty sandstone, and C is quartz. Fig. 18 (p. 485) shows a fault, the throw of which is about 7 inches, as seen in the 2400-foot level. The sketch explains itself.

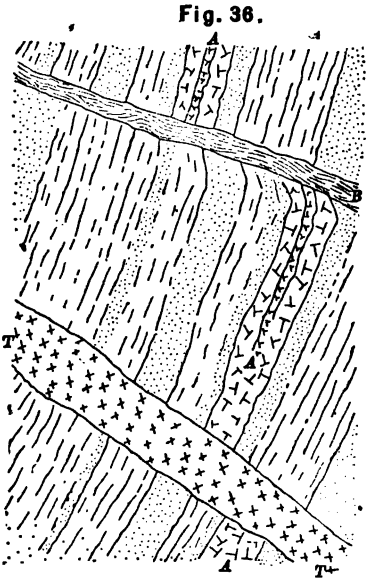
The underground workings of this very interesting mine* have uncovered five saddles, of which the one at the 2400-foot level is the third "payable" one. The other formations were cut at 560, 1560, 2000 and 2200 feet respectively. The east leg of the 2400-foot formation is the longest east leg in the mine, 40 feet. The longest west leg was worked from the 1600- to the 1870-foot level. The cap of the

* Mr. James Northcote, the mine-manager, gave me a great deal of this information and in many other ways assisted me in my examination of the gold-field.

560-foot saddle was the most payable portion of that particular formation. The east leg was short-lived, the quartz pinching, while the wall or "back" continued. Generally speaking, the west legs are the strongest and most auriferous in this, as in most of the saddles of the New Chum line of reef. No considerable body of gold-bearing stone has been worked below 1870 feet.

Barren quartz is rarely met with, but some is not of paying grade. From 1560 to 1870 feet was worked the great saddle-reef of the "180" mine, there being a large body of stone or ore of extraordinary richness on the west leg of the formation, which was first cut

Fig. 36.



Faulting Seen in the South End of the 1340-Foot Level in the Johnson's Mine.

at 1560 feet. The deeper levels show no change in the character of the country, and no large influx of water* hinders further sinking.

Recapitulation.

This concludes that portion of my notes which refers to the series of mines in which is to be seen the main saddle-formation of the Sandhurst of to-day. It may be taken as typical of the great runs of pay-quartz from which the field has obtained most of its gold. During the time (the four weeks preceding October 13th, 1890) in which I inspected the larger number of this group of mines, the returns were as shown in the accompanying table.

* The quantity of water hoisted is 3000 gals. per 24 hours.

Mine.	1st Fortnight, ending Sept. 27, '90.			2d Fortnight, ending Oct. 11, '90.			For the Month.			Depth.
	Loads.	Oz.	Dwt.	Loads.	Oz.	Dwt.	L'ads	Oz.	Dwt.	Feet.
New Chum Railway....	527	474	1	512	643	1	1039	1117	2	2025
Shenandoah.....	385	479	3	365	379	1	750	858	4	1990
North Shenandoah.....	209	95	10	145	100	4	354	195	14	1900
Young Chum.....	108	45	5	160	56	14	268	101	19	1400
New Chum Con.....	736	532	12	757	378	2	1493	910	14	1850
New Chum United.....	416	117	11	420	72	8	836	189	19	1850
Lansell's "222".....	.....	.....	.....	.....	.....	.....	98	88	0	1900
Lazarus Co.....	231	163	17	210	236	11	441	400	8	1960
Lazarus No. 1.....	256	52	7	249	138	13	505	191	0	1960
North Old Chum.....	181	143	11	279	251	10	460	395	1	2290

Ten mines are included in this list, there being no crushing from the "180" mine for this particular period. The returns for the separate fortnights are given so as to show how far the yields vary. Nine out of the ten are dividend-paying properties at the present time, and the figures show that the average yield is 444 oz. 16 dwt. from an average output for the month of 624½ loads,* these results being obtained from workings whose average depth is 1912½ feet from the surface. This is a good record for deep quartz-mining.

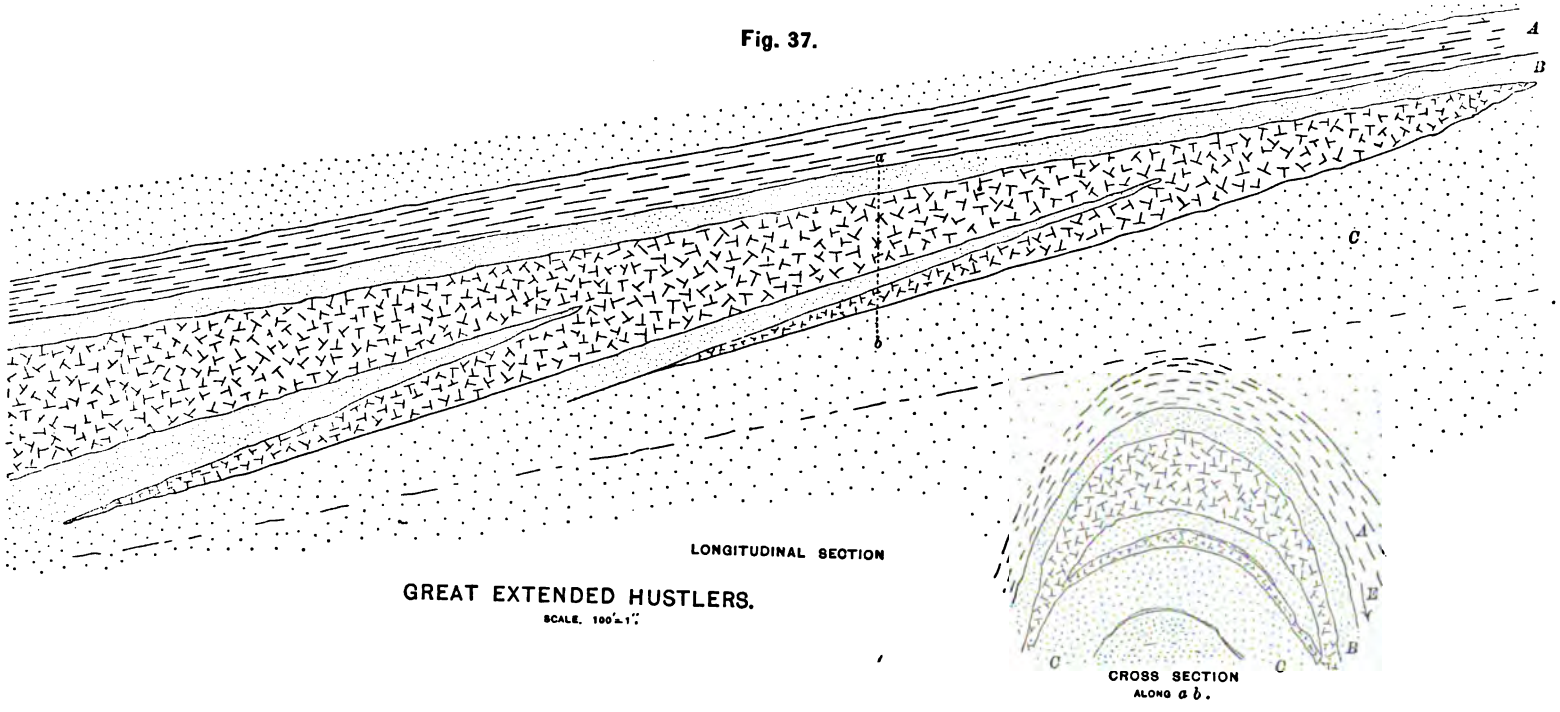
Other Underground Phenomena.

Before proceeding to describe the underground phenomena observed in others of the mines it will be well to refer again to the general structural geology of the district. Emmons† quotes a saying of the geologist Von Groddeck that the understanding of the true character of veins can only be arrived at by a study of the structure of the region in which they occur. Of no mining district can this be said more forcibly than of Sandhurst. As we have already seen, the gold-field is situated among the highly contorted folds of the Lower Silurian slates and standstones. The fissuring which is the accompaniment and result of extreme contortion has led to the production of very diverse forms of ore-deposition. We have seen something of the saddles (anticlines) and inverted saddles (synclines).

* A load is equal to 25 cwt. (avoirdupois, at 112 pounds) of ordinary ore, or 30 to 35 cwt. of concentrates.

† "Structural Relations of Ore-deposits," *Trans.*, xvi., 804.

Fig. 37.



LONGITUDINAL SECTION

GREAT EXTENDED HUSTLERS.

SCALE. 100′ = 1″.

CROSS SECTION
ALONG *a b*.

For reasons to be discussed in the sequel, the latter are neither so extensive nor so productive as the saddles proper. In addition to these two very beautiful types of ore-deposition, there are in the Sandhurst mines good examples of almost all the best recognized forms in which gold-quartz is known to occur. When the parting between two adjacent beds has been a line of movement, the distinct division so produced in the country is recognized and named a "back." It is usually accompanied by a smooth wall forming the surface of the harder of the two beds, and by more or less black selvage or gouge, resulting from the abrasion to which one of the two beds has been subjected. Such a "back" offers facilities for the deposition of quartz by the waters to which it gives a ready passage. A "back" accompanied by quartz, whether a mere thread or several inches in width, becomes a "leader," the word referring more particularly to the quartz seam which serves as a guide in prospecting the ground. Such a deposit would come under the category of "bed-veins."

Another very common, often also very extensive and profitable, formation is locally known as a "*make of spurs*." It consists of a network of quartz-veins, traversing a more or less definitely limited section of country. They frequently start in the neighborhood of a "back" or "saddle," and intersect the slate and sandstone at a strong angle with their bedding. These spurs are sometimes so numerous as to make the country almost entirely quartzose, while in other instances, though continuous and following a distinct belt of country, they may be so far apart as to hide their true character. In various forms these spur-systems constitute one of the most important sources of the gold obtained in the Bendigo* mines. When they accompany one of the main reefs they may be considered as "feeders;" when they form a distinct reticulation of quartz-veins they answer to the term "stockwerk."

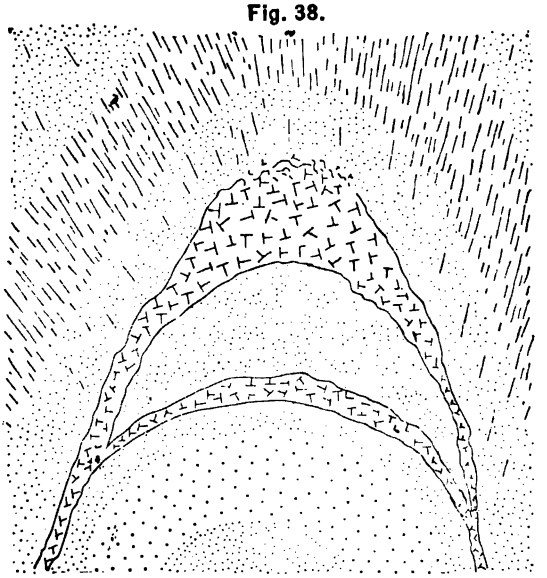
Midway between the forms known as "backs" and "makes of spurs" come the "lodes." These consist of occasional belts of country-rock, most frequently slate, which carry irregular seams of quartz, some of which are arranged parallel to the walls (the bedding planes of the country), while others have a transverse direction. Such occurrences are locally called by the generic term of "lode," differing from a "make of spurs" in that the quartz-seams are confined to one

* The best-producing mine at the present time—the New Red, White, and Blue Consolidated—is developing one of these "makes of spurs."

bed alone, and from the backs and leaders in being larger and less defined, and in carrying a greater proportion of country-rock.

Fissures containing quartz and traversing the country unconformably to the bedding, are often faults and would answer to the "true fissure-vein,"* a type which is of frequent occurrence in this field, though not so important a repository of the precious metal as the saddles or spurs. "Bulges," "blocks," etc., are names given to irregular bodies of quartz, presenting features more or less in common with the preceding types. At Sandhurst, as elsewhere, the different forms of ore-deposition cannot be arbitrarily labelled, exhibiting as they do frequent gradations from the one to the other. In the para-

Fig. 38.



Radiated Cleavage, Great Extended Hustlers Mine.

graphs which follow, I will describe instances of the type known as "saddles," leaving the other forms for subsequent consideration.

The South New Chum Mine.

At the southern end of the district—about 3½ miles from the group of big mines on Victoria Hill, some of which have been passed in review—there is a small mine called the South New Chum, which

* This form of ore-deposit was at one time erroneously considered as particularly favorable to the occurrence of continuous shoots of gold-bearing stone.

admirably illustrates the particular formation most characteristic of Sandhurst.*

This mine is working in regular "saddle-country." The longitudinal and cross-sections given in Figs. 19 and 20 (p. 487) will serve to explain it. The underground developments of this mine and its northern neighbor, have proved the existence of three formations, all pitching strongly to the south.

Formation No. 1 came to the surface 120 feet from the South New New Chum shaft. The line drawn in the section, Fig. 19, is the anticlinal axis of the saddle. Looking along the strike, it would appear like the ridge of a roof, sloping rapidly southward. At A the cap of this formation was strongly defined and very rich in gold. Near the surface both legs proved "payably" auriferous. At B the saddle was not "payable." At C the cap became gold-bearing, and was well defined as it approached the next shaft to the south.

Formation No. 2 is the one which is shown in the cross-section, Fig. 20, which is taken in a plane 5 feet south of the line of the shaft. Both the 170- and the 280-foot cross-cuts passed through it. At D, Fig. 19, the cap is broken, and there was no quartz in the legs. At A the cap became "payably" gold-bearing as it approached the line of the shaft. The east leg was not worked, and the west leg was also poor. At B there was a "payable" cap and west leg, between the 170- and 280-foot levels. At C the formation is irregular and broken.

Formation No. 3 is cut through by the shaft and is again partially exposed at the 550-foot level. The cap is irregular. The west leg is worked, but the east leg has not yet been reached by the cross-cut. Referring to the cross-section, Fig. 20, I will transcribe the notes made by me while underground.

At the 170-foot level.—East leg cut 10 feet east of shaft. The lava dike 12 feet and west leg 34 feet, both west of shaft. The lower portion of the cap is passed through in sinking at 125 feet, the top of the cap being 5 feet west of the shaft. The lava dike is 4 feet wide, divided by 2 to 3 inches of slate, which, throughout the upper workings, separates it into nearly equal portions. The cleavage of the country is with the east leg, slightly marked over the cap and more clearly defined a short distance above. On the other side of the saddle the cleavage is decidedly *across* the west leg. The formation pitches strongly southward, 1 foot in 6, and at times even 1 in 5.

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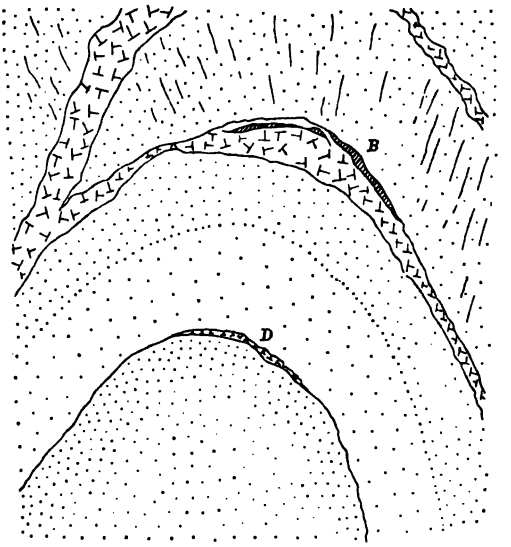
* I owe thanks to Mr. L. A. Samuels, the mine manager, who conducted me through the workings during my visits at the mine and gave much interesting information.

At the 280-foot level.—East leg 74 feet east of shaft. Lava dike in the plat.* West leg 84 feet west of shaft. The dike is increased in size, the two portions being divided by two feet of slate. The cross-cut shows a subordinate saddle underneath the main formation. This secondary saddle carries a little quartz (4 to 5 inches), which in the shaft showed "nice gold." The west leg (of the main saddle) consists of two parts, each about 5 inches in width, the quartz of which is very beautifully laminated or ribboned. The hanging-wall is clean and defined, inclining at a less angle than in the level above. The cleavage west of the center-country is slightly west, nearly upright, and cuts across the bedding, which is readily recognizable. The country east of the west leg is slate (2 feet thick) overlying sandstone, the thin bedding of the latter being very marked. The east leg as cut in the cross-cut is small and poor, but is overlain by country corresponding to that which covers the west leg.

Fig. 39.



Formation of Secondary Saddles, Great Extended Hustlers Mine.

At the 450-foot level.—The legs of the upper formation have not been sought for, as they both became small and poor below the 280-foot level. The east leg would be about 150 and the west leg 200 feet from the shaft. At 35 feet east of the shaft a western "back" is struck, this proving to be the continuation of a wall coming from a saddle below the level. The lava is cut at 36 feet from the plat; it here shows only a fine black line dividing the two portions.

At the 550-foot level. In the plat the sandstone of the west country shows ripple markings. The west leg of the third formation is cut close to the shaft, and has been driven upon for 118 feet southward.

The east leg has not yet been cut, though the cross-cut has been put out in

* The station at the shaft, often called the "station." Literally, "plat" means a flat place cut out. The Cornish talk of a "tip plat," the flat place where the ore is accumulated, with facilities for discharge below the level.

that direction for 69 feet. It is estimated that 20 feet more would find it. The cross-cut very clearly shows the bedding of the country which is pierced by the lava dikes, which here are 16 feet apart. The south drift is on the west leg. While the wall (the "west back" of the 450-foot cross-cut) continues its course, accompanied by a seam of quartz ("the leader"), the west "reef" turns off eastward and strikes against the lava dikes, but has not yet been crossed by them. The quartz which here is highly auriferous is found to occur both between the two dikes and west of them. The end of the drift is in lava.

Figs. 21 to 27 (p. 489) inclusive, illustrate some of the more interesting of the underground features already referred to. Fig. 21 shows the second formation as disclosed in the workings above the 170-foot level. There is no doubt as to the continuity of the beds forming this anticline. Sandstone A overlies the saddle, of which B and D are respectively west and east legs. C is the body of quartz forming the apex; above it there is a wedge-shaped mass of crushed country, consisting for the greater part of black broken slate containing a notable percentage of pyrites, but no "payable" amount of gold. The slate underneath the west leg varies from 2 to 3 feet in thickness, and is divided by a thin "bar" (3 to 4 inches) of hard sandstone. Underneath the east leg there is also from 2 to 2½ feet of slate, while below the cap this slate is irregular and forms "bulges." This bed overlies the hard sandstone H of the center-country, fractured and penetrated by spurs. The east back F, that is, the parting which forms the hanging-wall of the east leg, has been followed for 30 feet above the cap of the saddle.

Fig. 22 is the west leg, as seen at the 550-foot plat. The reef C is hard against the back A, which carries its own thin vein of quartz (the leader). D D are spurs from the west leg into center-country. These cross-spurs are a frequent feature of the country between the legs of a saddle.

Fig. 23 represents *in plan* the position of the "lava streaks" in the south drift of the 550-foot level. S and T represent the dikes; C is the quartz; and A A the "back."

Fig. 24 shows the relative position of the dikes, as disclosed in the 550-foot east cross-cut. The bedding of the country is very evident. T, which is from 18 to 20 inches thick, is just east of center-country, and has cut through it. S is 20 inches thick, and has just begun to shape its course with the bedding.

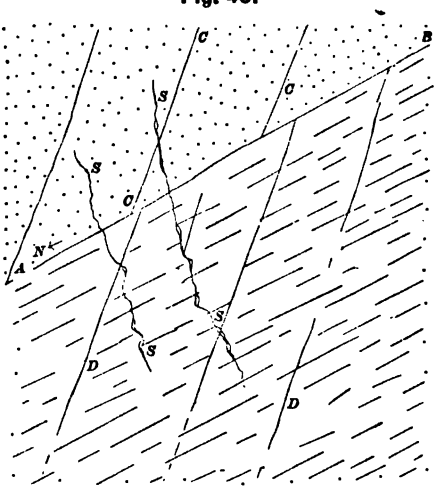
Fig. 25 represents the markings, natural size, in the decomposing lava of the west dike in the 550-foot cross-cut.

Fig. 26 shows the "west back," which, 40 feet above the 550-ft. level, runs into the west leg of an incomplete saddle (the third form-

ation). It will be understood that a "back" is such only so long as it follows the line of parting between two beds of the country; that when it follows a quartz seam it often becomes the hanging-wall of a reef; and finally that, before striking the quartz or after doing so, it may change its character and be a cross-fissure. In Fig 26, A is the back; C is quartz; S and T are the two portions of the lava dike which cuts through the formation. There is a fault, the throw of which is from 8 to 10 inches. The dike does not fault the quartz, but follows the line of an earlier dislocation. D is broken ground, traversed by spurs. B is an east back.

Fig. 27 is taken from the cross-cut at 190 feet, and indicates a local fault which throws the country about 3 feet east. B B is a

Fig. 40.



Bedding, Cleavage and Jointing, Great Extended Hustlers Mine.

bed of slate; F F is the line of fault, carrying a couple of inches of selvage and, on the under side, a little quartz.

This mine illustrates the general character of the saddle-reefs. To any one desirous of obtaining the key to the geological structure of the district I recommend the examination of formation No. 2 as cut by the 170- and 280 ft. cross-cuts, where the different features of cleavage and bedding of the enclosing country, the pitch and underlay of the quartz bodies, are all more distinctly marked than I observed them in any other mine. On the other hand the lower or No. 3 formation, as cut by the 450- and 550-ft. cross-cuts well illustrates the frequency with which the simpler type of ore-deposits becomes complicated almost beyond recognition. That this is due to

the action of faults, is evidenced by the slickensides on the walls and the crushed condition of portions of the country, such as the wedge of slate above the No. 2 saddle.

In this mine there is none of the perplexity occasioned in the deep mines on Victoria Hill by the difficulty in distinguishing the bedding from the cleavage, where the latter is highly developed. The cleavage which elsewhere so often obliterates the bedding, especially of the west country, is clearly distinguishable in the slate and sandstone beds disclosed by the different cross-cuts.

The 280-ft. level, where it is driven upon the western leg particularly, well illustrates this feature. The sandstone there seen overlying the reef is composed of a number of thin beds of different shades of gray rock, the light and dark laminæ are beautifully regular, and have a strong dip westward (with the reef), while the cleavage is only slightly off the vertical, giving the foot-wall in the "drive" the appearance of taking a roll. The side of the level had broken first along the wall of the reef—which is also the bedding of the enclosed country—and then along the nearly vertical cleavage.

In conclusion I may add that though the mine is called the South New Chum, having been originally supposed to be a southern continuation of that great "line of reef," it is now generally accepted as being on an extension of one of the "side-lines" to the west.

The Johnson's Mine.

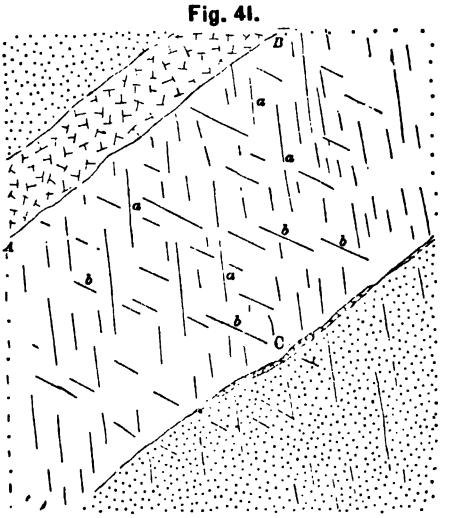
The finest example of the saddle-reef to be seen at the present time is in the Johnson's mine at Eaglehawk. This claim is not, like most of those hitherto described, on the New Chum line of reef, but on the northern extension of the Garden Gully, so called after the mine in which it had its greatest and richest development, the Garden Gully United. There has been but one saddle worked in the Johnson's mine, but that one has been exceptionally regular and rich. The apex is 980 feet from the surface; the west leg has been but little developed as yet, but the east leg still continues profitable beyond the 1340-foot level, which is the deepest in the mine.

All the gold obtained in the upper workings came from "backs" and "spurs," as well as other irregular bodies of stone or quartz.* Descending No. 2 shaft a thousand feet, one finds that the level is driven upon "the main east back," as it is called. It may be considered as an east leg which has no corresponding west leg; nor, so

* The Colonial talks of "stone," where the American speaks of "rock," and the Frenchman "minerai." Gold-quartz in Australia is "golden stone."

far as it has been followed, does it turn over to form a saddle. This "back" has been worked successfully for 300 feet along its strike, the quartz making in bulges, sometimes 100 feet in height, but of very variable longitudinal extent. The gold-contents varied from 3 to 10 dwt. per ton. These bodies of quartz pitch north. Before coming to an end they usually break up into a number of spurs, and they are generally to be found at those points where a number of small veins—feeders or spurs—join the main back. Such a "make of stone"* is to be seen above this 1000-foot level. The

Fig. 41.



Double System of Cleavage, Great Extended Hustlers Mine.

quartz, 12 feet wide, is crystalline and white, but well mineralized, containing vugs or geodes and occasionally patches of included country. This block of quartz pitches north, but 60 feet further is another which pitches in the contrary direction. A small regular seam usually accompanies the back. It is known as the "leader," and is distinct from the other irregular bodies of quartz which may be found along the line of the back. The terms "back" and "leader" are used interchangeably. The wall of the back is very well defined, and carries a couple of inches of black clay—the "dig"—which continues long after the quartz has died out. Figs. 28 and 29 (p. 491) will illustrate this. A is sandstone; C is massive quartz; B is slate; L is the leader; D is the dig or selvage; F F are small spurs. A cross-cut 85 feet east from this back brings us to the east

* A "make of stone" is the equivalent of a "shoot of ore" or an "ore-body."

leg of the big saddle. The cross-cut shows that on the hanging-wall there are 14 feet of sandstone, divided by three thin partings of slate. Both walls are very clean and distinct, the hanging particularly.

Above the stopes the saddle itself is seen. The appearance of the workings is very striking. The removal of the quartz has left a large vaulted chamber, over which the curving hanging-wall extends with the regularity of an arch of masonry, while the foot-wall underneath looks like the back of a boiler, the curve of the saddle being remarkably regular. As there is no timbering to obstruct the view, it is easy to sketch the heading. Fig. 30 (p. 491) shows the general structure of the formation. Looking along the crest—the summit of the curving under-wall—one can readily see that the pitch is to the north. The overlying rock is the hard sandstone which forms the hanging-wall; underneath comes 2½ feet of quartz; then follows a dark parting of slate, under which lies 12 to 14 inches of sandstone, then 5 to 6 inches of slate, mixed up with quartz, which latter gets rapidly smaller both east and west.

These layers of slate, sandstone and quartz are all conformable to the bedding of the enclosing country. The over-arching wall of sandstone shows wavy markings, a suggestion of the beautiful ripple-markings to be seen at the 1060-ft. level. There are also to be observed thin threads of quartz, in lines at right angles to the strike—probably following joint-planes.

The west leg has been followed only 30 feet on the dip. Fig. 31 (p. 491) illustrates its appearance. Under the hanging is 12 to 14 inches of clean quartz, C, well laminated. Underneath comes 2 feet of sandstone, A, separated by a black slate parting. Between this sandstone and the main foot-wall there is 5 inches of slate, B. The hanging is broken by feeders which carry gold. The cleavage of the country-rock is to the east, as indicated on the right of the drawing.

The east leg has supplied the major portion of the output of the mine, and has proved highly auriferous. Fig. 34 (p. 493) shows its appearance just below the cap of the saddle. The reef at this point is divided into two parts by about 12 inches of sandstone, H. The upper portion, C, is 2½ feet wide, mottled by the inclusion of bits of country, M, near the hanging-wall, and well laminated or ribboned (N) near the included sandstone, H, which last is cut up by spurs, S S.

The underlying slate, B, is black and graphitic. The lower por-

tion, E, of the reef is less regular, and lies directly upon the hard sandstone, D, of the main foot-wall. At about 40 feet below the cap the east leg, as seen in Fig. 32 (p. 491), pinches to 3 inches at C, with 4 feet of slate, B, and sandstone, A, between it and the hanging. The decrease in size is compensated for economically by an increase in the gold-contents.

At the 1065-foot level the lode has been worked out. At the 1280-foot level a cross course throws the east leg, which is here from 9 to 10 inches wide, eastward 6 feet, which is equivalent to its being moved from one wall to the other of the lode-channel in which it is situated. This is shown in Fig. 33 (p. 491). At this point a body of quartz, C, about 6 feet by 6 feet, is formed. It is very white, and has

Fig. 42.

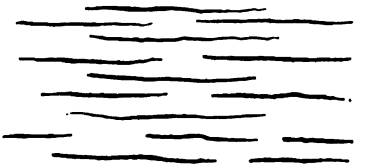


Fig. 43

Cleavage and Crystalline Lamination, Great Extended Hustlers Mine.

been known to carry coarse gold. A is sandstone; B, slate; R, the reef. The north end of the 1280-foot drive shows two faults* in the lode. See Fig. 35 (p. 495). The larger has a throw of 8 inches. Both dip westward, but at different angles. The quartz, A, on the foot-wall side, is much the best in appearance and in yield. The foot-wall itself is well defined and unbroken. B is broken, slaty rock, containing pieces of quartz. C is a body of quartz, white, and containing but little gold.

In the stopes the same faults are found closer together. The spurs

* In the mine they call them "slides." Elsewhere in Australia this term alternates with "heads" and "breaks." In the New Zealand coal-mines the synonym is "troubles."

in the foot-wall going west from the reef pay to follow for 20 or 25 feet. The quartz of which they are formed is white and the gold is coarse, not varying in this regard whether near or distant from the main lode. The tenor of the ore is such that the reef will pay to work when only 3 inches thick, the miners breaking with it feeders, spurs, mullock,* everything as it comes. The average width, however, is from 12 to 16 inches, at which size the lode is most gold-bearing,—better than when it increases to its maximum width of 3 feet.

At the 1340-foot level the east leg is still rich in gold. It is becoming more inclined, having travelled east 10 feet in the last 60, while between the 1000- and 1280-foot levels it moved 90 feet. There is not so much country-rock between the walls as in the levels above. The quartz is covered by a very clean striated hanging-wall, which is 4½ feet west of the main hanging of the lode-channel. In the breast of the north drift the reef is 6 feet wide. The foot-wall is a sandstone so far altered and hardened as to be a quartzite. Ripple-marks are beautifully distinct. There is a fine line of stopes above this level. Faults similar to those in the 1280 are seen. The broken country under the faults shows evidence of having been arranged in lines parallel to the lode. In the south drift is seen the lava dike, shown in Fig. 36 (p. 497). B is a fault, the course of which is marked by soft broken slate. The reef, A, is twice dislocated, the second time by a small fault, which has served as a passage-way for the dike, T.

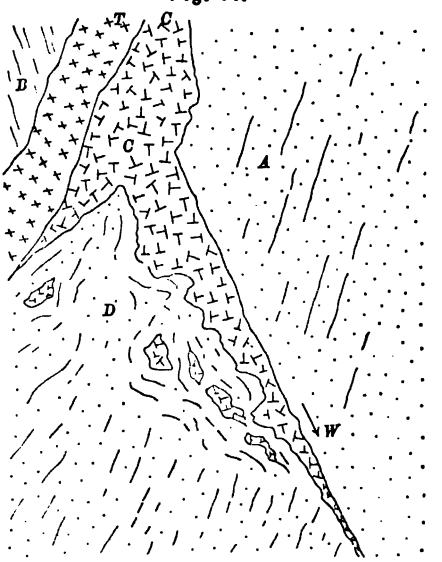
This 1340-foot level is at present the deepest in the mine. The reef has been followed by a winze for a further depth of 35 feet. The shaft is now being sunk so as to enable deeper levels to be opened, and so continue the development of the mine. A cross-cut is being driven to intercept the west leg.

The richest ore which this formation yielded was obtained just below the turn-over of the saddle, but very rich shoots were found irregularly distributed through the reef, having the same pitch as the formation itself. The quartz, speaking generally, is patchy, and sometimes extremely rich bunches are found. One of these lately gave 150 ounces of gold from a half-sack of quartz.

* Mullock is waste rock. A "mullock-tip" is brother to the western "dump." Originally, mullock was the name given to the basalt which covered the deep leads of Ballarat; but since then the alluvial diggers have introduced the term into quartz-mining; and at Sandhurst it refers to the slate and sandstone which is broken with the quartz of the reef.

As we have seen, most of the workings are on the east leg, which, as observed in the levels underneath the cap of the saddles, presents all the appearance of an ordinary quartz-lode with walls parallel to the bedding of the enclosing country. The cross-cuts show the bedding. The alternating slate and sandstone are called "bars" by the miners. Thus, one hears of a cross-cut having just passed through a "tight bar of sandstone." Even where the difference in the nature of the rock forming adjacent beds is very slight, there are lines of parting (usually seams of slate) which indicate the stratification.

Fig. 44.



False Saddle, 1670-Foot Level, Great Extended Hustlers Mine.

The cleavage is distinct and uniformly to the east at an angle of 60°.

The Johnson's mine for the half-year ending June 30, 1890, paid dividends amounting to £19,441 2*s*. The yield of 12,367 loads of stone was 9146 oz. 3 dwt., worth £38,639 5*s*. 3*d*., and averaging 14 dwt. 19 gr. per load. The following figures will be of value as indicating the tenor of the ore at the different levels during the half-year:

Part of the Mine.	Loads.	Yield. Oz.	dwts.
Above the 550-foot level, . . .	121	13	10
" 1000-foot level, . . .	1886	991	3
" 1130-foot level, . . .	289	206	16
" 1200- and 1210-foot levels, .	2156	1274	17
" 1280-foot level, . . .	5145	3276	17
" 1340-foot level, . . .	2113	2803	19

The average yield from the stone at the bottom of the mine was 1 oz. 7 dwt. per load.*

The Great Extended Hustlers Mine.

The Johnson's mine is on the Garden Gully line of reef, but the next to be described is on the third of the three great anticlinal axes—the Hustlers. The Great Extended Hustlers is one of the "record-mines" of the district,† and at the present time exhibits a formation of subsidiary saddles which is both peculiar and interesting. Fig. 37 (p. 499) shows the formation as seen in the workings between the 1700- and 1800-foot levels. The cross-section is taken along *a b*, and, with the longitudinal section, will explain itself. A is slate, B and C are both sandstone. It is seen that the anticlinal axes (the ridges of the saddles) pitch strongly to the north, the main axis 1 in 5, while the secondary saddles pitch 1 in 3. The main formation has been worked for 800 feet in this mine, and extends for some distance into the adjoining claims. The secondary saddles‡ have a length along their strike of 250 and 300 feet respectively.

In Fig. 37 (p. 499) it is seen that, commencing at the south, the right hand of the drawing, a cross-section would show one large saddle; as we proceed north the great width of quartz becomes divided by a horse of sandstone which, like a wedge, eventually splits the formation into two saddles, having at *a–b* a width through their caps of 40 and 12 feet respectively. At this point the top of the main cap is irregular, broken into a mass of spurs. The foot-wall of the lower saddle forms a very clean arch. Further north again, the lower saddle gradually pinches out, while simultaneously the main formation increases in size, very soon to be again divided by another wedge of country-rock, with a repetition of the conditions already noted in the previous case.

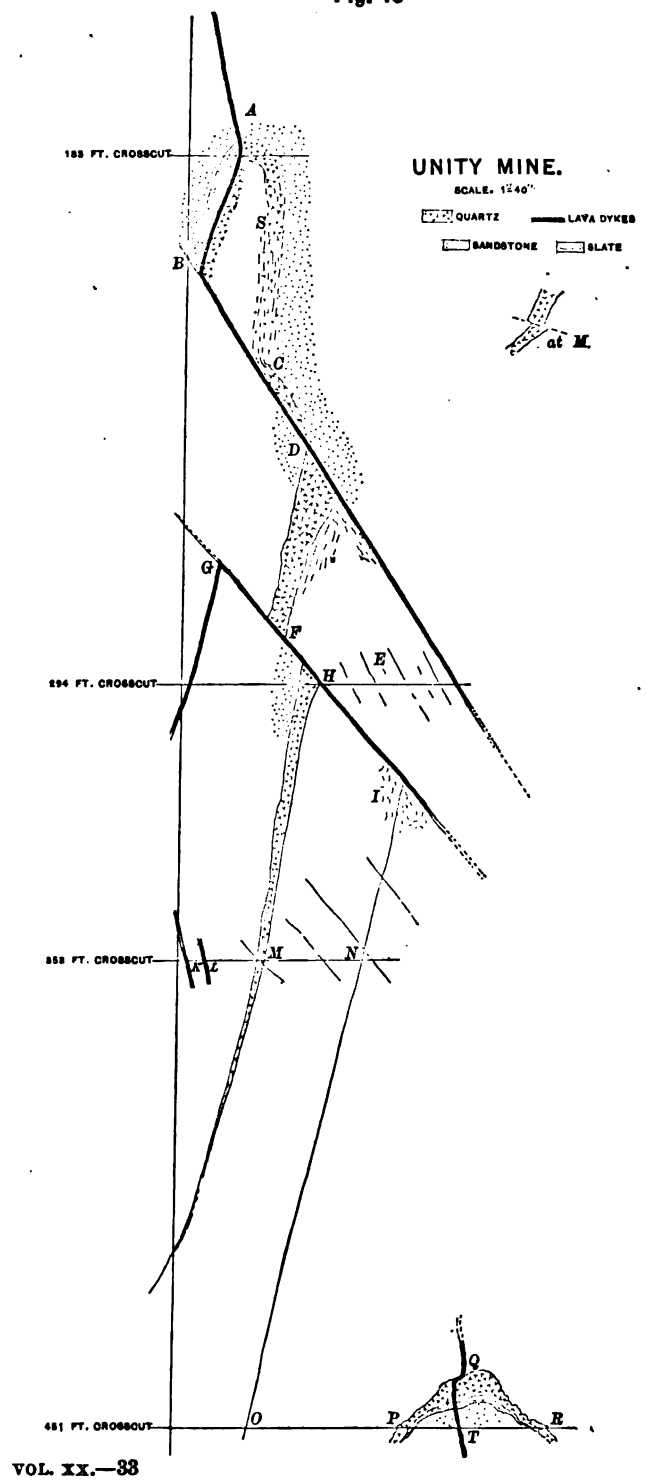
Before the two saddles become divided by the country-sandstone,

* My thanks are due to Mr. Williams, the mine-manager, as also to the underground foreman, for many of the details here given.

† It is a noteworthy fact that the three mines which perhaps rank first in the returns of gold which they have made, should be on the three main "lines of reef," and opposite each other, illustrating the old mining aphorism, "ore against ore." I refer to the "180" on the New Chum, the Garden Gully on the line of the same name, and the Great Extended Hustlers on the Hustlers line.

‡ During my later visit to Sandhurst the manager informed me that he had come across a third of these subsidiary saddles. I had not the time to inspect it. I have here an opportunity of thanking the mine-manager, Mr. Thomas Weekley, for the valuable assistance he gave me in visiting this mine, as well as for his guidance in examining the old workings of the Unity mine.

Fig. 45


UNITY MINE.
SCALE. 1″=40′
QUARTZ
LAVA DYKES
SANDSTONE
SLATE
at M.
A
B
S
C
D
G
F
E
H
I
K
L
M
N
O
P
Q
R
T
185 FT. CROSSCUT
294 FT. CROSSCUT
353 FT. CROSSCUT
451 FT. CROSSCUT

the main body of the quartz begins to show included fragments of country; becomes, as the Sandhurst miners say, "mullocky."*

At the 1750-foot level, the east leg carries three or four inches of quartz against a sandstone wall which exhibits transverse markings. These will be referred to later. The quartz is crushed so as to resemble common table-salt, and occasionally the surface where it has not been pulverized shows beautiful slickensides, polished to the likeness of ivory, and veined with black streaks due to the grinding of the pyrites. The west leg shows a large body (5 to 6 feet) of white and splintery quartz.

The saddle was most auriferous at and immediately below the commencement of the legs. This was particularly marked in the case of the west leg, which was by far the more auriferous of the two. In another formation, worked in the levels immediately overhead, the reverse was the case; the east leg paid best, often proving continuously gold-bearing for from 150 to 200 feet; whereas the east legs are generally short-lived.

The sandstone overlying the saddle is 10 to 15 feet in thickness, and is overlain by a bed of slate 25 feet thick over the cap, and penetrated lower down by the 1800-foot cross-cut, the slate being thinner on the west side than on the east, where it is much disturbed and broken.

The sandstone over the cap has a nearly vertical cleavage; that of the west country is strongly east, while that of the eastern country is slightly westward; in other words, the cleavage is radiated. This is illustrated in Fig. 38 (p. 501).

In Fig. 39 (p. 503), one of the lower secondary saddles is shown. The cleavage over the cap is also radiated and distinct, though it does not entirely hide the bedding. Above the cap, at B, and in the quartz itself, there are a few inches of black slate. In the center-country underneath, there is the suggestion of a third saddle, D, in the quartz seam following the slate-parting which marks the bedding of the sandstone.

Fig. 40 (p. 505) is taken in the 1805-foot level. A B is a bedding-plane, sandstone above and slate underneath. The lines C C are joint-fractures; S S are small spurs cutting across both divisions in the country-rock. Underneath the second saddle in the same level are seen the fractures shown in Fig. 41 (p. 507); A B is the east leg, C is a quartz seam following the bedding; *a a* are lines of normal cleavage.

* That is, mixed with mullock or country-rock.

Fig. 46.

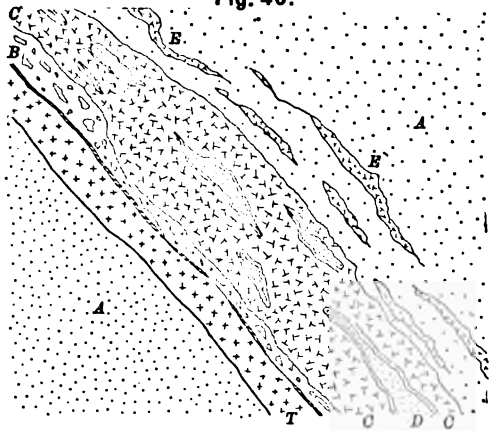


Fig. 50.

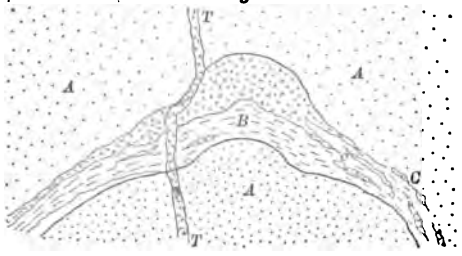


Fig. 47.

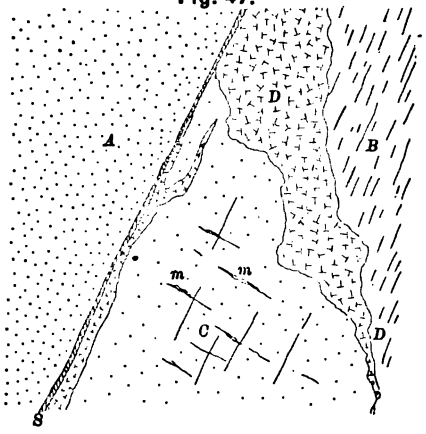


Fig. 51.

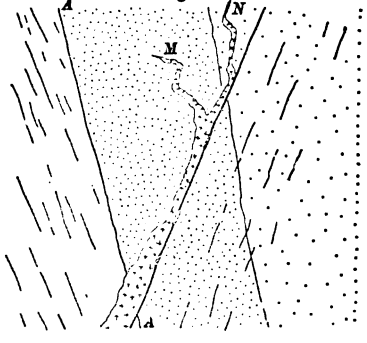
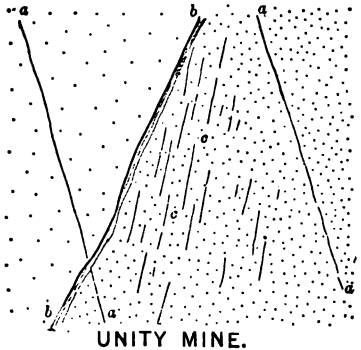
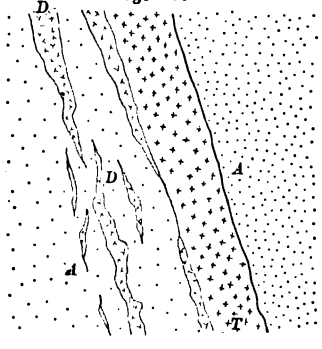


Fig. 49.



UNITY MINE.

Fig. 48.



The system of fractures to which *b b* belong appears to me to be also cleavage, but due to the transverse (north and south) folding of the country. In other words, there is here double cleavage, one due to the east and west, and the other to the north and south, bending of the beds. The planes *b b* are at right angles to the plane of A B (the bedding) and are not confined to one bed only.

Reference has been made to some peculiar markings seen above the east leg of the main formations. They are more plainly seen in the sandstone above the west leg at and about the turn-over of the saddle. As seen underground, they present the appearance shown in Fig. 42 (p. 509). The deepening and thinning of the dark lines is due to the unevenness of the surface produced in breaking it. Actually, the lines are as shown in Fig. 43 (p. 509). They are identical with the cleavage, and present an instance of cleavage which is also crystalline lamination, that is, they are lines due to compressive strain, along which a dark indeterminate mineral has been developed and arranged.

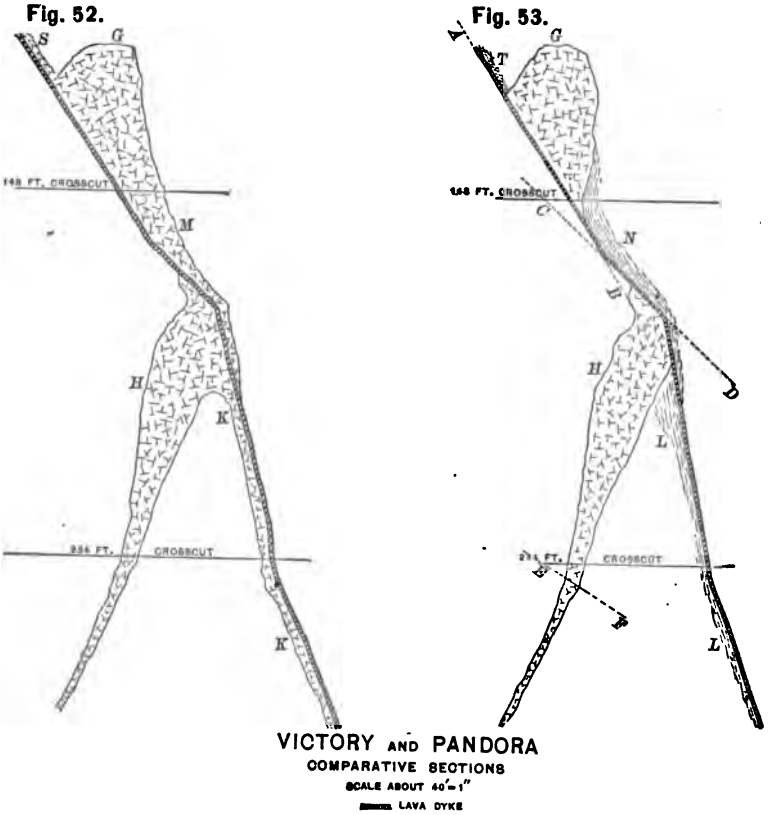
Fig. 44 (p. 511) is a "false saddle" seen in the 1670-foot level. B is slate, the cleavage of which is indicated, as is also that of A, which is a slaty sandstone. T is a lava dike, 12 to 14 inches wide. D is a slaty sandstone, broken up and containing irregular fragments of quartz like C. The bedding, to the west, is indicated by the arrow. This is a typical instance of a quartz formation which is very apt to be mistaken for a true saddle.* Quartz is formed along the fissure which cuts across the bedding of the country, the bedding-planes themselves also show quartz, and the point of intersection with the fissure may be marked by a large body of ore. In this case the dike follows the line of the cross-fissure and a formation of quartz (C) is seen. It is to be noted that the line of bedding is more favored in the deposition of quartz than that of the fissure which crosses it. Such "makes of stone" are notably irregular and uncertain. At the point where the sketch is taken the lava is just west of "center-country," and the reef underneath the lava is of very limited extent. In going south, the dike makes over to the east country, the country dipping east, and the accompanying quartz then becomes more continuous.

The Unity Mine.

At the Unity mine, which is on the Garden Gully line, there is a good example of the extensive faulting which frequently obtains in this district, making difficult the development of the ore-bodies.

* As already noted, p. 480.

Fig. 45 (p. 513) is a section, 40 feet to the inch, which will serve to explain the occurrence. B A C is an imperfect but true saddle, since the country dips east and west with the two legs. There is no true east leg, but rudiments of it are seen in the spurs which traverse the slate at S. Sandstone overlies the cap and the west leg, as well as the bed of slate in which are the spurs which replace the east leg. At 28 feet below the cap, the country is faulted, the throw bringing the west leg, the only part of the formation which continues well-



defined, from B to D, a distance of 35 feet. Another slide at a further depth of 38 feet (along the dip) causes another break, this time of 8 feet only. Again at M there is a dislocation, which has not gone further than to cause a derangement in the straight course of the lode. We will follow the formation through the different changes which it undergoes. At the 183-foot cross-cut, where it is first seen, the top of the cap has been stoped away, offering to view the partial

formation of the arched structure, which is so beautifully distinct in some of the mines, the Johnson's for instance. The west leg is strongly defined, and yielded well at the time it was stoped. The spurs which replace the east leg were worked for a depth of 25 feet, and eventually merge into the formation shown at C in Figs. 45 and 46 (p. 515). In the latter, A is the sandstone of the overlying country; E E are spurs; C is quartz, which includes several fragments of country at D D; B is a layer of brecciated quartzose material containing gold, whose origin was probably the fragments of quartz broken away by the fault from the west leg up above. All this overlies the lava dike T, which is from 6 to 8 inches wide, and is separated from the underlying sandstone by the lower wall of the fault-fissure. The whole formation, inclusive of the portions C D B, may be considered as forming the slide or fault which at a later date became a passage-way for the dike. The west leg continues, well-defined and easily recognizable, for a depth of over 200 feet, in spite of its being three times dislocated. From D to F it forms a large body of stone,* decreasing from 12 to 5 feet in width, which yielded good returns, many crushings giving 5 oz. per ton.

At the 290-foot level at the east end of the cross-cut, the first fault is seen. A drift 15 feet long shows at its opposite ends the appearance illustrated in Figs. 47 and 48 (p. 515). Fig. 47 shows the south end. The sandstone, A, is over the line of fault, which along its course has a clean wall carrying a few inches of soft selvage, S. The sandstone, C, underneath, is considerably fissured, and in the fissures are the small quartz-seams, *m*, *m*. The slate, B, shows cleavage parallel to the fault. Quartz is formed along the main wall, and a large spur,† D, follows the bedding of the country. At the other end of the drive, only 15 feet from the south face, the heading presents a totally different appearance, as is shown in Fig. 48. A is sandstone; T is the dike which, in the plane of section, Fig. 47, must have pinched out temporarily. D D are quartz-veins parallel to the fault. The country through which it cuts dips east.

The 291-foot cross-cut gives a section of the lode and of both slides. The lower fault is 28, and the first, or upper, 57 feet east of the shaft. The second fault has more dip, but, like the first, it shows a bulge of lava at one end and none at all at the other end, 6

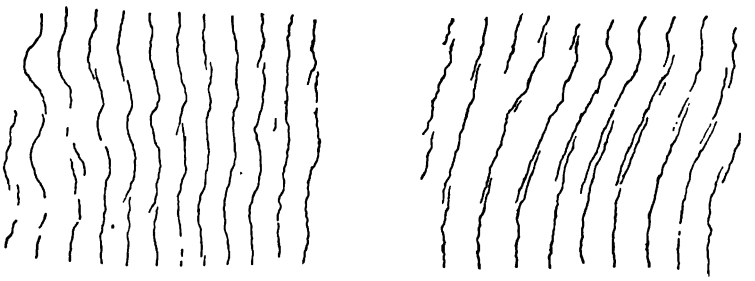
* That is, "mill-stuff," "quartz," or "dirt," as it is variously termed by the miners in various parts.

† Were it to continue for any distance it would be called a "back," since it follows the line of bedding.

feet distant, of a raise which has been put up to cut the lode. The country between the two slides is much disturbed, and shows lines of fissure in sympathy with the faults. It is full of flat spurs, having a slight dip eastward. The country west of the faults contains no spurs, but it is hard sandstone and much fissured. At 15 feet west of the second fault or slide, the country begins to show a westerly dip. In Fig. 49 (p. 515), taken at this point, *a a* are fissures parallel to the faults; *b b* is the bedding, indicated by a slate parting between two layers of sandstone; *c c* is cleavage, nearly vertical.

At the 353-foot level (see Fig. 45, p. 513) the quartz has decreased to 2½ feet, but it is still a strong lode. Neither of the faults is to be seen, as the cross-cut is not far enough east. At M there is seen an attempt at faulting which has gone so far only as partially to dislocate

Fig. 54. Fig. 55.



RIPPLE MARKS.
JOHNSON'S MINE. 1065 FT. LEVEL.

the reef. At N there is a "back," a line of parting between two beds, parallel to the lode, which goes up as far as the second fault, where a formation of spurs is found on both sides of it. This is the last we see of the reef, which decreases rapidly in size and becomes unprofitable, the dip taking it across the shaft before reaching the level of the 451-foot cross-cut..

This cross-cut intercepts at O, Fig. 45, the back which we saw at N in the 353-foot cross-cut. It is well-defined and continues under foot. The cross-cut ends in a very well-formed saddle, P Q R, which is shown in greater detail in Fig. 50 (p. 515). The outline is suggestive of a double contortion. The cap makes a very clean curve. Sandstone, A, overlies the quartz, both being penetrated by the dike, T. Underneath the quartz is slate, B, in which are the spurs at C, which replace the east leg. The west leg is divided into two portions, separated by slate. Going south, they are wider apart. The

slate under the cap shows lines of bedding, and its division from the underlying sandstone forms a clean, very beautiful arch.

The lava dikes, which form so marked a feature of the section, Fig. 45 (p. 513), have many points of interest. A dike (varying from 12 to 20 inches in size) cuts through the cap at the 183-foot level. It has a slight eastern dip which, on striking the quartz, is changed for the dip of the west leg, through which it passes (it is now 6 to 8 inches wide) until meeting the fault, which it then follows, as the line of least resistance. It underlies the formation at C, and is last seen in the 294-foot cross-cut.

Another dike (4 inches only in width) comes down, keeping company with a "leader," cut near the shaft at the 294-foot level. It is probably a branch from that which follows the No. 2 fault, as seen at H, Fig. 45. This latter dike is probably again seen in the lava which strikes at Q against the saddle, following the west leg for a few feet, and then cutting across the quartz to continue its course through the sandstone of the center-country.

At the 353-foot level, near the plat, is seen the dike shown at K, which is also illustrated in Fig. 51 (p. 515). There it is seen that it cuts across the bedding (A A), following a fissure, and that it appears to die out in small threads at M and N.

About 5 feet east of this point there is another dike, shown at L, Fig. 45, which is 9 inches thick, and is continuous so far as it is seen.

The two faults encountered in this mine are found again in the neighboring mines. They get wider apart to the north. In the Carlisle they are divided by 40 feet of country, while southward they approach each other, and in the Victory and Pandora Company's ground they are seen to cross. I saw a section of the upper workings of the Pandora in which these same faults are readily recognizable, though they had been misinterpreted by the draughtsman. In Figs. 52 and 53 (p. 517) are given two comparative sections. Fig. 52 is a sketch reproducing on a smaller scale the old drawing to be seen in the office of the company. Fig. 53 indicates what appears to me to be the real situation. In Fig. 52, S is called the leader, G the cap of the saddle, H the west leg, and K the east leg of another more complete formation. The lava dike follows a slide which has an irregular course through the two bodies of quartz. Just below the 234-foot cross-cut the legs are bent.

Compare this with the section of the Unity. The two mines adjoin, and we know that southward the two faults or slides are ap-

Fig. 58.

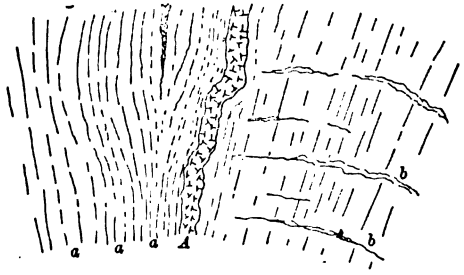


Fig. 59.

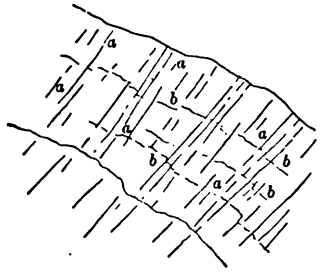
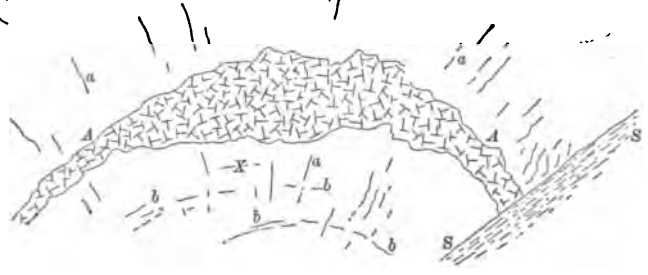


Fig. 56.



LAZARUS MINE.
1250 LEVEL

Fig. 57.

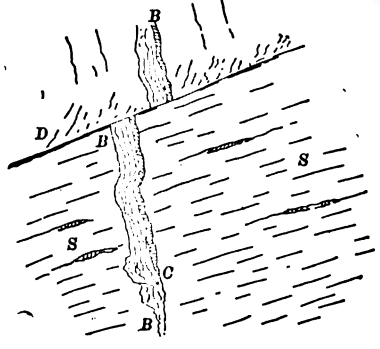
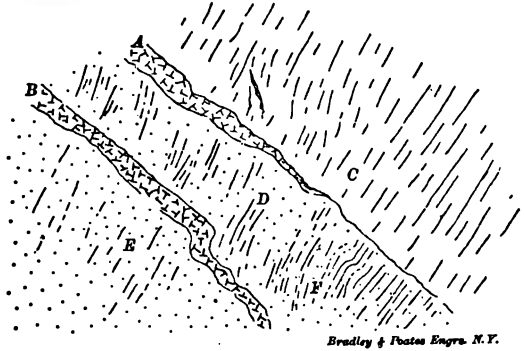


Fig. 60.



Bradley & Foates Engrs. N.Y.

proaching each other. Now refer to Fig. 53, where G is the cap of a formation, which, like that of the Unity, of which it is undoubtedly an extension, has no east leg, while the west leg, H, is faulted by the two slides, A B and C D. In the Unity we saw that the lower fault was flatter than the other—it is so here, where they cross. Below the 234-foot cross-cut we have a partial dislocation due to the fault, E F, similar to that seen at M in the Unity (Fig. 5, p. 513). The portion, N, of the formation corresponds to that shown at C in the Unity section, while the quartz which accompanies the dike at L L is similar to that which usually follows the line of the fault, as indicated in Figs. 47 and 48 (p. 515.)

The question arises, which way was the movement? The only evidence in reply is that obtained by noting which part of the country-rock is most shattered. In the Unity the country between the faults has been fissured, that west of them less, and that east of them more. That the west country has not been subjected to much movement is shown by the regularity of the formation in the 451-foot cross-cut. It seems to me, therefore, that the movement was an upward one on the part of the country east of the first fault, accompanied by a lesser movement in the same direction by that which lay between the two faults, decreasing westward until at M it had almost died out.

Other Indications of Structure.

Having given a description of some of the typical mines of the district, it remains for me to add such other evidence gathered in the various mines as bears most directly upon the structural relations of the ore-deposits.

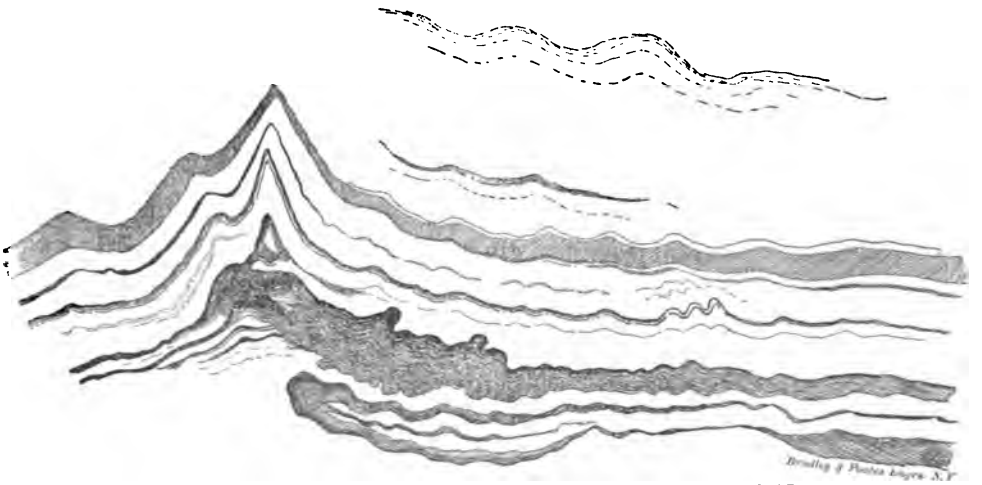
Of the evidence offered by the underground workings, none is more beautiful than the ripple-marking at the Johnson's mine. This is seen at its best in the 1065-foot north level. Figs. 54 and 55 (p. 519), are sketches made underground to illustrate it.*

Fig. 54 was taken 5 feet north of Fig. 55. The crests of the waves are 3 to 3½ inches apart, the rock in which they occur is a very hard sandstone, shading into a quartzite, and forms the footwall of the reef. On breaking the face the markings cannot be followed into the next layer of sandstone, this being due to the metamorphism which the rock has undergone, and its cementation along the lines

* It requires photography to do justice to this very interesting occurrence. All I can do is to give roughly an idea of this striking evidence of the bedded character of the reefs.

of original deposition. The preservation of the ripple-marks is due to the fact that the sandstone face which shows them was overlain by a soft slate, which has since in part been replaced by the quartz of the lode. The markings can be followed for 100 feet in height, and for more than 200 feet in length. The reef of which this is the foot-wall dips east about 70°; it was in this part of the mine richly gold-bearing, and it was found in breaking the quartz* that it broke with difficulty, owing to the resistance offered by the projections of the corrugations which were so striking a feature of the stopes.†

Fig. 61.



CONTORTIONS IN SANDSTONE.
LAZARUS. 1250 LEVEL
¾ NATURAL SIZE.

These ripple-markings were in east country; those which I saw in the South New Chum were in west country. At the 550-foot plat in the latter mine, immediately behind the "west reef," there is a

† The ripple-marked wall formed "the shooting-wall," that is, it was utilized in blasting as the line along which the quartz detached itself most readily.

‡ Soon after my third visit to Sandhurst, I saw an almost exact duplication of these markings in the sands of to day. It was in New Zealand. We were trawling by moonlight; and while pulling in the net I noticed that the sand forming the bottom of the shallow water in which we stood was covered by a series of ripple-marks, whose distance apart and little irregularities resembled those which I had seen in the Johnson's mine; but these on which I stood were of to-day, those were laid down in the estuaries of the Silurian seas; these were soft and yielding, while those had been consolidated into a quartzitic sandstone, now one thousand, and at one time many thousand feet below the surface.

bed of sandstone whose *under* side shows a *cast* of ripple-markings, in other words instead of getting ridges, you get hollows, the true ripples on the underlying bed are not to be seen, but the sandstone which was deposited upon it still retains the impression, the negative of the face of sand which it overspread.

At the Lazarus mine is found the formation shown in Figs. 56 to 61 (pp. 521, 523), reproduced from sketches taken in some old stopes at the bottom of a winze leading from the 1250-foot east level. Fig. 56 gives a general view of the saddle. A A is the cap, which is not at all regular in outline, and both above and below breaks up into spurs. The east leg is cut off by a fault, on which the winze was sunk. The manager informed me (the ground is filled up now), that in stoping, it was found that the west leg also was cut off by this same "slide."

The remarkable feature of this formation is the development of radial cleavage. In the drawing, *a a* are lines of cleavage, *b b* are structural lines parallel to the bedding. The bedding is for the most part completely obliterated by the cleavage. On careful search the lines of original deposition can be traced, but with difficulty and only partially. The most striking evidence, however, is that of some extremely delicate but distinct markings found in the sandstone underneath the arch of the saddle, markings which prove in miniature the contortion to which the rock has been subjected. They are shown ½ their natural size in Fig. 61 (p. 523),* and their position as regards the saddle is indicated by X in Fig. 56. The rock in which they occur is a very fine-grained sandstone, the contortions being rendered distinct by the alternation of dark- and light-gray laminæ.

Fig. 57 was taken in the winze. The slide S is 5 feet wide, and the slaty materials of which it is composed are arranged parallel to the bounding walls. A small quartz-seam B is faulted,† the throw being about four inches. It is accompanied by a narrow band of black slaty filling. The edges of the beds of the country are bent and broken at their contact with the walls of the "slide"—at D.

Fig. 58 is taken in center-country, under the saddle. A is a vein of quartz, a spur, filling one of the many fractures produced in the bending of the beds; *a a* are lines of cleavage, at this point very marked; *b b* are quartz-seams arranged along the structural lines,

* This might be taken as the cross-section of a mountain range; and certainly these foldings, though only covering a few inches, are typical of the plication which has produced mountain ranges.

† A fault within a fault.

which are parallel to the bedding. This may be called "secret bedding," since the fractures are doubtless in sympathy with the bedding and along lines of original deposition, only now made visible by the fissuring through which relief was obtained at the time of the plication of the rocks.

Fig. 59 is in east country, just above the slide or fault. In Fig. 60, C is slate; D, slaty sandstone; E, sandstone. A and B are quartz-spurs thrown off by the east leg. At F the cleavage is twisted.

This formation, the several portions of which have been briefly described, throws a good deal of light upon the structure characteristic of saddle-reefs.

Evidence of the structure of the district can be obtained without

Fig. 62.

Surface-Cut, Victoria Hill.

going underground, since large open cuts are numerous along the main lines of reef. Several sketches made in the surface-excavations are shown in Figs. 62 to 66, inclusive. Figs. 62 and 63 (pp. 525, 527) were both taken behind the engine-house of the "180" mine on Victoria Hill. Fig. 63 shows where the quartz has been for the most part removed, leaving an arch overhead. C is *débris;* D, slaty sandstone; while A is a lava dike, the same dike which is repeatedly seen underground and which forms one of the most striking features of the deepest workings, as shown in Fig. 14 (p. 485.)

Fig. 62 is about 100 yards south of Fig. 63, and shows the southern continuation of the same anticline. At B, the bedding is very distinctly to the west, the rock being "corduroy," which weathers very slowly. The country about the apex of the saddle

has undergone considerable alteration, and only traces of the original bedding are now to be seen.

Fig. 64 (p. 529) is taken in the vicinity of the New Red, White and Blue mine. The saddle is very sharp and distinct. C is an old drift, while D consists of material which has been filled in.

Figs. 65 and 66 (pp. 531, 533) will appear curious. They are sections of the same anticline, taken from the opposite sides of a narrow cutting. The space covered is only 7 to 8 feet in width.* The wavy lines are marked by small seams of quartz which follow the contorted bedding in the country-rock. The nearly vertical veins are interesting as indicating the fractures which took place along the axis of the fold.

This closes the evidence which I offer as bearing most directly upon the structural relations of these very remarkable ore-deposits of Sandhurst. Out of the large number of notes and sketches made by me while underground, I have endeavored to place before the Institute those which seemed most directly pertinent. Instances in which geology, particularly structural geology, and practical mining are so closely allied can rarely be found. The close relation which they bear to one another is hardly understood by the busy underground managers of the mines, save in a few very noteworthy cases; and this is scarcely to be wondered at, seeing that almost without exception the references to and the descriptions of the Sandhurst mines, meager as they are, on the part of geologists, engineers, etc., are quite incorrect; their authors having confounded the bedding with the cleavage. Misled by the cleavage, "its mysteriously deceptive harmonies with the stratification"† on the east side of the saddles, and its often complete obliteration of the bedding on the west side, they have considered that the east leg did conform to the bedding, but that the western cut across it, that is, the ore-deposits were "saddles" in name only.

There is one noteworthy exception, namely, a short note which appeared in the quarterly report of the Mining Department of Victoria, for December, 1888, by E. J. Dunn, formerly government geologist of the Cape Colony. In that note attention is directed to

* There are several large open cuts behind the Victoria quartz mine, and having noticed that in different parts of the workings the country dipped east and west, I hunted for the turn-over of the saddle. It was only after a second search that, in leaving the old excavations by a narrow road-cutting, I found what is represented in my sketches. Recent rain had rendered the structure beautifully distinct.

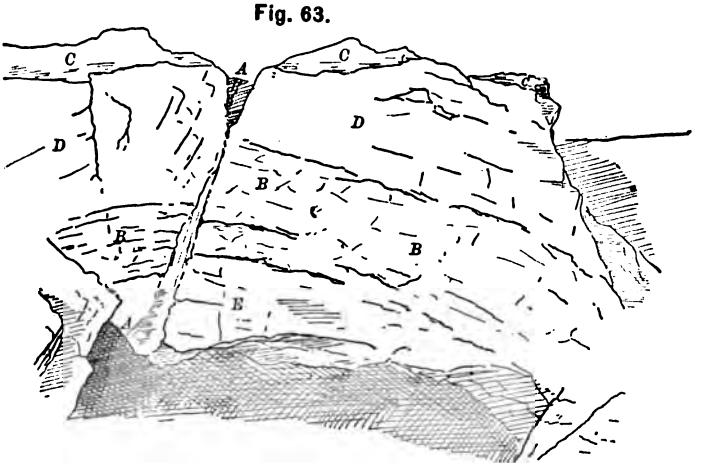
† Ruskin, in *Schisma Montium, Deucalion.*

this point. Had it come under my notice previous to going to Sandhurst, I should have been spared much of the difficulty which I experienced before the true nature of the deposits became manifest.*

That the typical ore-deposit of the Sandhurst gold-field is a true anticline there can be no doubt. As already pointed out, the saddle is not the only formation in which the gold-quartz is formed, since the spurs, backs, lodes, etc., have at all times been responsible for a very large share of the output; but the saddle is the distinctive form to which the others are for the most part accessory and subsidiary.

The ripple-marks frequently observed underground are incon-

Fig. 63.



Open-Cut, Showing Saddle, Victoria Hill.

trovertible evidence of the original position of the beds. That, for instance, the east leg of the big saddle in the Johnson's follows the true line of stratification, is amply proved by the very beautiful markings seen in the 1060-foot level. Similarly, ripple-marks are seen on the faces of beds dipping west. Other no less conclusive though less striking evidence has been given in the descriptions of the various mines.

Mining is a commercial business and not a scientific pursuit. The following of a lode on its dip ceases whenever the quartz which it

* In visiting Sandhurst one usually goes first to the more famous mines on Victoria Hill, such as the celebrated "180" mine and others near it. It so happens that in these mines the structure of the country-rock is most hidden, because the cleavage is most highly developed. The smaller mines at the other end of the field show the true character of the ore-deposits far more clearly. Among the latter the South New Chum has been described here.

contains is no longer auriferous; and as a consequence the mine-workings rarely disclose the ultimate limits of the lower portions of a saddle-formation. Lower cross-cuts will intercept a thin but pronounced parting in the country, a "back" which could be traced upward until it formed the boundary of a big lode or "reef;" or they may cut through a "spur," which if followed would be found to lose itself in an east or west leg, the distinction in the two cases being solely that of conformity with the bedding of the country-rock. Where the legs, as parts of a saddle-formation, end, and where they become one or other of the various types of cross-fissure, no man can say. It may be considered that whenever they lose their true character as bedded veins they cease to be the lower portions of a saddle; but the distinction is theoretical and arbitrary; for the gold knows no difference and is found in both.

The same system of fracturing may produce in one part of the mine a "big reef," a leg of a saddle which is highly auriferous, while overhead* in another part it may be a mere fissure, carrying no quartz, and lined only with a thin seam of black clay, which cuts across the country overlying the saddle. Again at a lower depth, while still retaining its characteristics as a portion of a saddle, it may carry quartz which is not sufficiently gold-bearing to work, or it may slowly degenerate into a "leader," if the quartz continue, or a "back" when even that has ceased, becoming nothing more than a parting between a bed of sandstone and a bed of slate. Again, it may eventually cross the bedding, cease to be a true "leg," and become a gold-bearing "spur"; and finally it may be traced into a mere thread of white quartz, one of many, traversing a bed of sandstone.

To make nice distinctions would be arbitrary and out of accord with the facts; the same fissure in one mine plays many parts.

While, therefore, I would not say that the legs of a saddle-formation never go astray from their correct course between the bedding planes, such an occurrence would be found only when they are beginning to die out, and does in no way modify our view of the true nature of the "saddle-reef," which, notwithstanding interesting local variations of structure, is a distinct type of ore-deposit—essentially traversing the Silurian slates and sandstones in conformity with the bedding, and forming along the anticlinal undulations of the enclosing country those bodies of gold-bearing quartz which are the most pecu-

* See Fig. 21 (p. 489), where an instance of this kind is shown in the South New Chum.

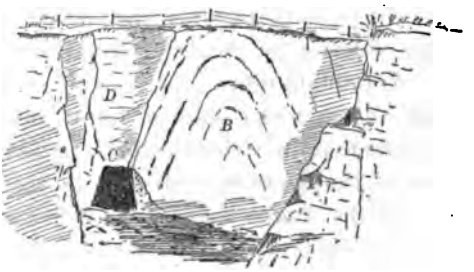
liar feature of the mining industry and the most valuable depository of the wealth of the Bendigo gold-field.

Mine-Management.

Fig. 67 (p. 535) is a view of the New Chum line of reef from Eaglehawk, copied from the quarterly mining report of the Victorian government.

There are at present engaged in the exploitation of the mines 143 companies and 21 tribute-parties, the former including four private mines. The gold-field has been developed by the energy of its own people, the amount of outside—chiefly Melbourne—money invested being comparatively small. Sandhurst has profited greatly by the enterprise and success of one or two mine-owners, such as the late J. B. Watson and the present owner of the "180" mine; but its development has been retarded, on the other hand, by the prevalence

Fig. 64.



Surface-Excavation, New Red, White and Blue Mine.

of one of the worst systems of company-financiering. The actual management of the mines, both above and under ground, is in the hands of capable and careful men; but they are hampered by the fact that reserve capital is almost unknown. Of the 28 dividend-paying companies of 1890, four appear also on the call list. That is to say, these four companies both levied calls and disbursed dividends during the year. A series of fortnightly sixpenny dividends may be followed by a number of fortnightly calls, supplemented by a bank-overdraft—a state of affairs which may swell the volume of business and keep a number of clerks employed, but which, to say the least, is quite unbusinesslike and highly prejudicial to the proper development of mines which require extensive and systematic exploration. The fortnightly dividend is the bane of Bendigo. Whenever a rich saddle-formation is discovered, all haste is made to open it up; as soon as the cross-cuts and drifts have intersected the ore, stoping is at once commenced, the quartz being broken away as

rapidly as possible, while all development-work is for the time practically suspended. The results of a few crushings enable the directors to pay off the bank-overdraft and enter upon the regular announcement of dividend after dividend, at intervals of a week if a fortnight seems too long! No part of the profit is laid aside as a reserve-fund, and, as a consequence, when the ore-body is worked out, the dividends abruptly cease, and the bank-account is overdrawn to pay running expenses, until it is necessary to call upon the shareholders for fresh assessments. This foolish round of alternating surplus and deficit, of squandering, borrowing and taxing, has gone on year after year, notwithstanding some of the more thoughtful managers are fully aware of its harmful character. The result may be seen in the cases of mines which divided in profits a few years ago sums varying from £100,000 to over £250,000, and are now almost idle for want of funds to open up new ground. The immediate distribution of profits leads to a continual change in the *personnel* of the shareholders, so that a mine when poorest may be owned by men who had no share in the property when it was most productive. Mining is thus brought down to the level of mere speculation—an evil not unknown, under different forms, in other districts, and certain everywhere to cripple the legitimate industry of mining.

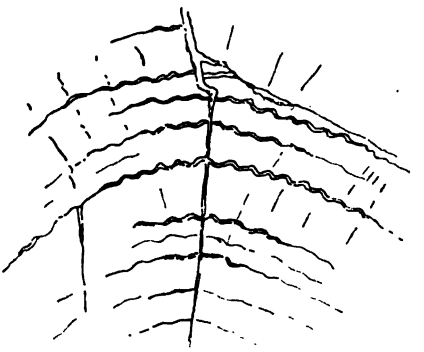
The management of the mines is controlled by the directors, the "legal manager" and the "mine-manager." Since, as a rule, these are all resident in the vicinity, and exercise more or less direct supervision, it is largely a question of personal influence which is in the ascendant. A mine-manager, as known to English companies, that is, a manager of all the mining work, or a superintendent, as known in the United States, is not found on this gold-field. The responsibility of superintendence is much scattered. The "mine-manager" receives the pay of a foreman—about £20 or $100 per month. The "legal manager" is the business man and book-keeper. The directors control the policy of the company, and also do much of the work often delegated elsewhere to a consulting engineer. Such a thing as a trained mining engineer is not to be found upon the staff of any of the companies; and the work done, as a consequence, lacks progressive character. The actual mining work under ground leaves nothing to be desired. It is carried out under the eyes of men whose superiors as miners cannot be found; but a want of system is to be observed in the development of the mines.

Treatment of the Ore.

The handling of the ore in the mills is far from satisfactory. The explanation is simple. Of the managers of 27 mines which I examined, the names of 25 indicated the fact that they hailed from the west of England, while the other two, though native-born, were of the same stock. The men of Devon and Cornwall are miners second to none; but they make the most unprogressive of mill-men.

The gold-ore of this field belongs to the simplest type of the free-milling class. As a rule, it consists of white quartz carrying a very small proportion (from 0.25 to 1 per cent.) of arsenical pyrites. Iron and copper pyrites, blende and galena are also frequently present, but in very inconsiderable amount. The gold varies greatly in its mode of occurrence, but is usually coarse, in the rich ore

Fig. 65.



Anticline with Curious Markings, near the Victoria Quartz Mine.

almost invariably visible, and of uniformly high purity, being worth £3 17*s.* 0*d.* to £3 19*s.* 0*d.* per ounce—pure gold being worth at the London mint £4 4*s.* 11½*d.*

The quartz itself is usually dead-white, with an appearance which, in California, would be considered most unfavorable to the presence of gold. It has a splintery fracture, rendering it easy to break. The mill-stuff consists of the quartz mixed with from one-third to over one-half of vein-filling and country-rock, which varies. The ratio in amount of slate to sandstone and the consequent hardness of the material to be crushed is constantly changing. It is the fragments of excessively hard quarzitic sandstone which cause most wear and tear to the milling-machinery.

Stamp-milling of the simplest kind is the method employed to extract the gold. The mills are constructed on the spot by excellent

local foundries, at a price usually far below what is paid in the western United States. A first-class stamp-mill of 40 heads, engine and shed included, with shaking-tables but without rock-breaker or automatic feeder, can be contracted for at from £6000 to £7000 or about $30,000. The battery-frame is generally of iron. Whether this mode of construction is advantageous to the work of the mill is open to doubt. Both the breaking and the feeding of the ore are done by hand, and in this respect the mills of Sandhurst, in common with most of the colonial batteries, are a standing disgrace to the modern mining industry. Inside plates are not used. Plain, not electro-silver-plated, copper plates are used to arrest the gold on amalgamating tables. Blankets precede the shaking-tables, which have the end-shock, and are a variation of the much-used Rittinger type.

The stamps weigh from 800 to 950 pounds per head; drop 8 to 9 inches, usually 70 to 75 times per minute, and crush, as a rule, about 2 tons per head per 24 hours. The depth of discharge or issue varies widely, but the average is about 3 to 3½ inches. The screen or grating is made of round-punched Russia iron, having 120 to 180 holes per square inch. The bullion is from 940 to 960 fine.

The percentage of concentrates varies from ½ to 2 per cent. per ton of ore milled. These are treated at the local "pyrites works," of which there are several; the method of extraction varying from arrastra to chlorination. The charge is generally £3 per load of 25 to 30 cwt.

A discussion of the milling-practice in detail and from a technical standpoint would be interesting and instructive; but I must content myself in the present paper with the above indication of its general character.

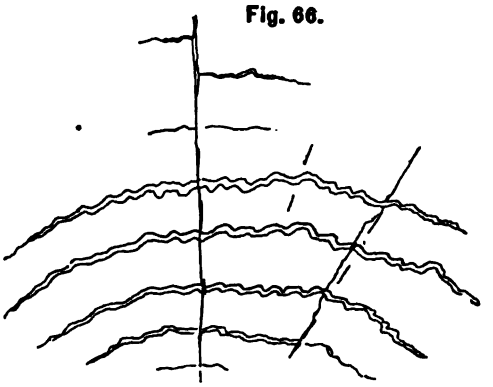
Government Assistance.

The serious decline in the output of gold in Victoria has led to the introduction of government assistance to mining. Whether this course has done good or harm is a vexed question. In the hope of giving renewed vigor to the mining of the colony, the Victorian government has distributed for several years sums amounting to £80,000 per annum, the intention being to encourage exploration. A miner, a group of miners, or a company needing funds to proceed with a cross-cut, a level or a shaft to cut a known lode, or to explore for a suspected one, makes application before a local committee, elected by the miners, and called the Prospecting Board. This body

reports it favorably or otherwise to the Secretary for Mines. Out of a large number of applications submitted, the most promising and deserving are selected. The terms of the grant are "pound for pound;" that is, for a grant, say of £500, there must be another £500 furnished by the applicants, making £1000 available for the particular work (and that only) designated in the application.

Such prospecting-grants are now generally condemned. Beyond question the system has completely failed to bring about fresh discoveries by prospecting. I have seen its operation in different parts of the colony, particularly in the mountainous districts which have been, as yet, least prospected; and the result appears to have been only to pauperize mining. It is always understood, that should the subsidized party succeed in placing the assisted mine in a dividend-

Fig. 66.



Anticline, from Opposite Side of same Cutting as Fig. 65.

paying condition, the amount of the grant should be repaid to the government from the first profits. This has been done in instances so few as to be remarkable.

On the other hand, this annual dole has done harm by encouraging that improvidence, whether of companies or of individuals, which is the chief hindrance to systematic mining. It has long been a bad feature of colonial mining, as I have shown above, that no portion of the profits is put aside as a reserve to carry on the work of development during lean years; and this prospecting-grant encourages that evil. Having been too eager for dividends to provide for the time when the reef might "pinch out," the mine-owners fall back upon the fatherly assistance of a benevolent government. Mining is a business; and if a mine has sound prospects of success it is very rarely indeed that the money is not forthcoming to carry on the work of development. The money thus distributed could be far better

employed in the endowment of a good central mining school, to replace the second-rate technical institutions which at present usurp the name.

Prospecting with the aid of the diamond-drill may be said to form a part of the scheme of governmental assistance.

During the past year, some interesting work in this direction has been carried out at Sandhurst. At the invitation of Mr. George Lansell, who offered to provide the necessary compressed air and other facilities, the Victorian mining department decided, toward the end of 1889, to put down a series of bore-holes from the bottom of the celebrated "180" mine, which had then reached a depth of half a mile. The machine used was Leschot's patent, manufactured in San Francisco, the core being $1\frac{1}{8}$ inch in diameter. Four holes were put down, having an aggregate length of 2335 feet. The record of each was as follows:

No. 1. Bored horizontally eastward from a point in the 1300-foot level 197 feet east of the shaft. Length, 683 feet. Commenced January 16, and finished March 17, 1890. Alternating beds of slate and sandstone were intersected with occasional thin seams of quartz. No information of any value was obtained, and the drilling was stopped by a strong flow of water, preventing the sufficiently rapid removal of the sediment from the bit.

No. 2. Bored downward to a depth of 453 feet, with an inclination of 9 inches in 16 feet, from center-country in the 2600-foot cross-cut. The ground varied greatly in hardness, and finally became broken up, when water, having a temperature of 97° F., and under a pressure of 110 pounds per square inch, was struck.*

* While the diamond-drill was at work, a good deal of notice was drawn to the heat of the water encountered in boring at the bottom of the "180" mine. The thermometer showed an average of 97° F., and at one time a maximum of 107° was reached, as recorded by three instruments. This was at a depth of 450 feet below the 2600-foot level or 3050 from surface. This temperature is by no means surprisingly high. The average annual mean temperature at Sandhurst is 60.2° F., and allowing for an increment of 1° for every 55 feet, we should expect at 450 feet a temperature of about 115° F, which agrees fairly well with the facts as disclosed in the "180" mine. However, the matter was considered of sufficient interest to induce a thorough examination by the government analyst, Cosmo Newberry, of the contents of the water so obtained. To this end, two and a half tons of the water was evaporated down to 10 gallons, care being taken to keep out impurities, and 3125 grains of solid matter remained, mostly carbonate of iron. Assays were made, but no gold was obtained, though a trace of a metal of the platinum group was given by the precipitated material. The results were disappointing, in failing to throw light on the occurrence of the gold in the quartz formation. The annual mining report of the department says that a further (optical) examination is to be made.

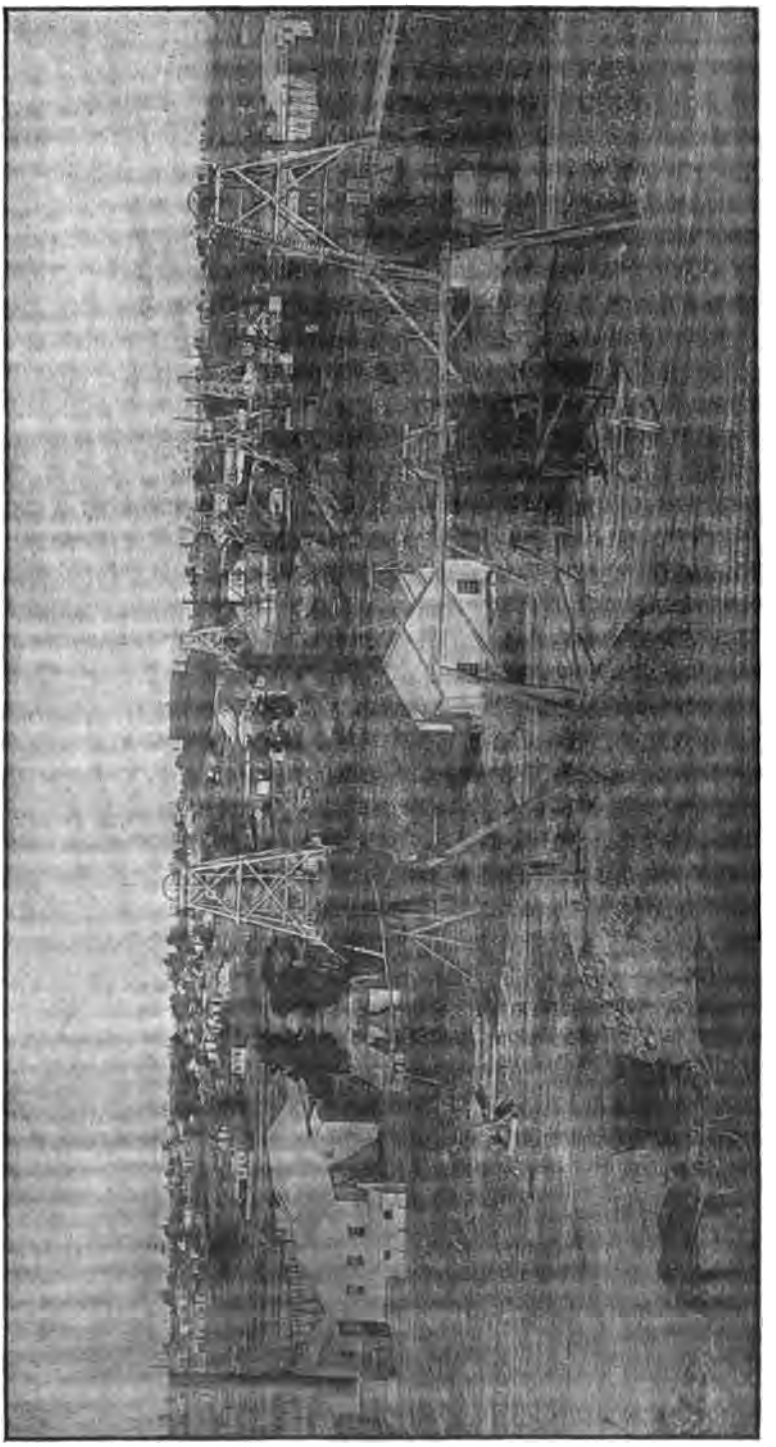


FIG. 67.

View of New Chum Line of Reef, Bendigo.

No. 3. Bored at a slight inclination below horizontal for 636 feet westward from the plat at the 2500-foot level. It intercepted several veins of quartz, the largest of which, however, was only 9 inches thick.

No. 4. Bored nearly vertical (inclining only slightly to the west) at a point 30 feet west of No. 2, in the 2600-foot cross-cut, to the depth of 562 feet below the level, or 3162 feet below the surface. It passed through about 200 feet of sandstone, as well as several beds of quartzitic sandstone, and cuts several series of small quartz-veins. Hot water under considerable pressure interfered with progress, as in the case of No. 2, though the temperature in this instance was lower.

The result of these borings was practically *nil*. No discoveries of importance were made; and it is doubtful whether the drilling had even a negative value in proving the absence of auriferous quartz in that portion of the country which it was attempted to test. If relied upon as a prospecting instrument under such conditions, the diamond-drill is likely to do serious harm by causing the condemnation of large areas of what may be good mining ground. In strata possessing regularity and continuity, such as the coal-seams; in the determination of the thickness of certain beds overlying known deposits, such as the sheets of basalt over the "deep leads" of California and Victoria; in the tapping of old workings which are under water; in the measurement of large ore-bodies (as was done in the case of the great bonanza at Virginia City); and in many other instances, no doubt, the diamond-drill, if judiciously employed, is a most useful aid to the miner; but in quartz-mining in general, and in such districts as Bendigo in particular, it is often likely to do irretrievable harm by fostering delusive hopes on one hand or by unnecessary discouragement on the other.

To support this view, it is not necessary to go outside Victoria. In the discovery of coal-seams and the exploration of deep beds of auriferous alluvium, the diamond-drill, under the judicious direction of the government geologist, has been very successful; but it has proved a very mixed and doubtful blessing indeed to quartz-mining. In the Sandhurst district, the peculiar structure of the "saddle-reefs," and the comparatively short extent of most of the bodies of gold-bearing quartz, are extremely unfavorable for tests of this character. This is particularly true of horizontal holes, which might penetrate the immediate neighborhood of rich formations without discovering them. If the drill is to be used at all, the

dangers attendant upon its use should be minimized by planning a series of holes close together, so as to test thoroughly and satisfactorily, at least a small portion of ground.

Dimensions of Claims.

Among the interesting features of this gold-field is the small area of the claims. The following table gives the areas of eleven contiguous properties on the New Chum "line of reef." The claims are of very irregular shape (see Fig. 7, p. 477), and in some cases the length of the tract is much greater than the figures given, but for purposes of comparison the measurement is taken along the supposed line of the New Chum reef.

Areas and Depths of a Group of Mines, Victoria Hill, Bendigo.

Name of Mine.	Area of the Property.	Length along strike of reef.	Depth of Shaft
	Acres R. P.	Yards.	Feet.
Ironbark Quartz	4 1 36	220	2140
Victoria Consols	22 3 15	259.6	2162
Gt. Central Victoria	10 1 17	250.8	1970
Victoria Reef Quartz	7 0 25	140.8	2302
Lansell's "180"	14 1 12	171.6	2641
North Old Chum	2 1 25	88	2310
New Chum and Victoria	13 3 12	110	2300
Old Chum	11 2 32	110	2208
Lazarus Co. Lazarus No. 1	22 1 7	122 122	2173
Lansell's "222"	16 0 19	99.4	2100

It will be seen that the average area of each property is but a little over 11 acres; that one of the claims, having an area of only two acres, has workings over 2300 feet deep; and that the ten shafts belonging to eleven companies have an average depth of 2230 feet, and are distributed over a total distance of less than a mile along the strike of the lode. The small area of these properties is primarily a relic of the early days when the claims consisted of twenty yards "along the line of reef" for two men. A certain amount of consolidation followed on the initiation of deep mining; but this was limited to the immediate needs of the time, and has long since been

outgrown by the very extensive exploration of more recent years. A circumstance permitting the working of these small areas is the absence of any large quantity of water in the mines. In the particular group of properties cited above there is not a single pump. The little water that finds its way into the workings is raised with tanks alone. At the "180," the deepest mine in Australia, 3000 gallons are hoisted per 24 hours.

The largest areas held by mining companies are as follows:

	Acres.	Rods.	Perches.
Hercules and Energetic,	85	3	17
Johnson's Reef,	71	2	21
Catherine Reef United,	65	3	12
New Red, White and Blue Cons.,	41	3	27
Garden Gully United,	41	3	0

It is seen that even the largest properties are of a size which would be considered small in many districts. It must be admitted, however, that the holding of an extensive acreage does not, in Sandhurst, necessarily mean extensive explorations; since, by reason of the want of working-capital, most of the companies confine their operations to such portions of their holdings as are in the immediate vicinity of the main shaft. This bad feature of the mining work will be slowly overcome, no doubt, by further consolidation. The twin Lazarus Companies have set a good example by sharing one working-shaft; and a recent attempt to consolidate the North Old Chum and the New Chum and Victoria, though unsuccessful, may be taken as an indication of the growth of a proper appreciation of the necessity for such measures.

Deep Mining.

The list of deep mines just given suggests the general subject of deep sinking, a feature of mining at Bendigo to which reference is often made in the colonies. This practice owes its beginning to the "180" mine. When the mine was purchased in 1873, the shaft was only 400 feet deep, but the neighboring North Old Chum, New Chum and Victoria, Old Chum, and Lazarus, were doing well on a reef which traversed the mines from 600 to 700 feet below the surface. The sinking of the shaft of the "180" mine was commenced. At about 600 feet £120,000 of gold was won. Sinking was continued in the face of much ridicule, and slowly the mine attained a depth which in those days was considered very great. Much money was expended without any return, until, in 1883, after ten years of

steady development, the top of the saddle was cut in the 1548-foot level. An ore-body of extraordinary richness was uncovered, and stoping began; but the sinking of the shaft continued. The Melbourne *Argus* of that day said: "The success of this venture in deep mining decides the prosperity of Sandhurst for a further decade at least." The mine is now 2641 feet deep. The success of deep prospecting having been proved in this trial-instance, imitators were not wanting; and there were soon several mines competing for the record of greatest depth. Deep sinking developed into a craze which seemed at one time likely to do serious injury to the proper exploitation of the mines. The result is seen to-day in the 18 shafts, each of which has a depth of over 2000 feet.

Among the causes of this state of things the structure of the ore-deposits may be named first. The Bendigo miner has an *embarras de richesse* in the matter of quartz-formations. Should a saddle be intercepted, the first crushing from which yields scarcely enough gold to pay expenses, it is soon left on one side in the search for the next formation, which is usually not far below. *Below*, not *ahead*, for, instead of longitudinal extension, the miner on this gold-field looks for a vertical repetition of the auriferous formations. Did each mine uncover one lode only, there would be more extensive and patient prospecting at any one given level; but, as matters stand, the exploration is not so thorough as it should be, for the reason that it is divided among several levels.

The most potent cause is, however, seen in the success of the deep mines. Deep sinking has not drawn blanks alone, but many prizes, such as the *bonanzas* of the Great Extended Hustlers, at 1800 feet; the New Chum Consolidated, at 1810; the Shenandoah, at 1900; the Lazarus, at 2000; Lansell's "222," at 2000; the New Chum Railway, at 2025; and the North Old Chum, at 2290 feet. As I have said, there are 18 shafts over 2000 feet in depth. They belong to 20 companies, there being two instances—the twin Shenandoah and the twin Lazarus companies—where one main working-shaft is used in common. Out of these 20 companies, 7 are upon the dividend-list; or, to inquire further, 4 are working quartz which is highly auriferous and 6 are breaking ore which is yielding profits, while of the remaining ten, nine are in that process of prospecting and development which here succeeds (though it ought to accompany) a period of stoping, and one is idle. This is a record above the average of even shallow mining.

The actual yield, during 1890, of the deepest-producing mines is

given in the following table, based upon the quarterly report of the Victorian Mining Department:

Name of Mine.	Quantity crushed.	Yield.			Average.			Depth of Stopes.	Depth of Shaft.
	Tons.	Oz.	Dwt.	Gr.	Oz.	Dwt.	Gr.	Feet.	Feet.
Gt. Extended Hustlers...	16,612	3,780	4	0	0	4	13	1,800	2,040
Lazarus Company.....	6,232	5,059	0.	0	0	16	6	2,000	2,110
Lazarus No. 1.....	7,282	3,665	16	0	0	10	1	2,000	2,110
Lansell's "222".....	3,454	2,510	2	0	0	14	12	1,950	2,105
New Chum Con.....	18,721	5,153	14	0	0	5	12	1,800	1,850
New Chum United.....	9,797	2,243	10	0	0	4	14	1,900	1,940
New Chum Railway.....	7,325	10,371	17	12	1	8	8	2,025	2,078
Shenandoah	7,474	5,039	3	0	0	13	11	1,990	2,010
North Shenandoah.....	1,048	948	3	0	0	18	2	1,990	2,010
Shamrock.....	2,496	1,446	19	21	0	11	5	1,800	1,840
North Old Chum.....	1,752	1,169	13	0	0	13	8	2,290	2,310
Kentish	3,117	1,183	13	0	0	7	14	1,800	2,113
Totals.....	85,310	42,571	15	9	...	.....	.....	23,345	24,516
Averages.....	.....	.....	...	...	...	9	23	1,945	2,043

Leaving on one side numerous small lots of ore, and taking into account only the results obtained from 12 of the deepest mines which were working regularly, it is found that during the year these dozen claims yielded 85,310 tons of ore, giving 42,571 ounces or an average of 10 pennyweights per ton, the average depth at which the ore was broken being 1945 feet.

All but one of these mines are on the dividend-list. We must inquire further in order to arrive at a just appreciation of the significance of this analysis. Of these 12 mines, 11 are paying dividends on ores broken at an average depth of about 2000 feet. There are a similar number, working at an equal average depth, which are not at present making profits. It is most probable that, during an interval of a year fully a third of the former will step into the ranks of the latter; that is to say, they will cease to break highly auriferous quartz and become prospectors for a new run of ore-ground, while, on the other hand, an approximately similar number will replace them from the list of mines which at present are in process

of development. We see, then, that at the present time half of the mines working at depths between 1800 and 2600 feet are profit-paying. The average yield of their ore is 10 dwt., while that of the district as a whole is 9 dwt. 5 gr. per ton. The best returns at present come from the New Chum Railway, which, at 2025 feet depth, is breaking quartz averaging nearly 1½ ounces per ton. During the year this mine paid dividends amounting to £21,672 on a paid-up capital of £28,589 and a nominal capital of £36,690. Its total dividends to date have been £52,078.

During the second half-year the Shenandoah, working at 1990 feet, produced 4555 tons of quartz, yielding 3717 oz. 17 dwt., or an average of 16 dwt. per ton. Dividends of £8400 were paid during the 6 months, on a called-up capital of £12,800. Altogether this company has returned £59,600 in dividends.

The Lazarus Company, working at 2000 feet, on 16-dwt. ore, paid out during 1890 dividends of £15,750 on a paid-up capital of £63,187. The total dividends of this mine to date have been exactly the full amount of its nominal capital, £67,500.

The results obtained in some cases appear to leave a very slight margin of profit. The average of the Great Extended Hustlers, the New Chum United, and the New Chum Consolidated mines is very low. It speaks well for the size of the lodes and the handling of the ore, that dividends are possible at a depth of nearly 2000 feet on 5-dwt. mill-stuff. The New Chum Consolidated is an instance of very successful gold-mining. The return of 5 dwt. 12 gr. shown in the list given above is higher than the ordinary average of the mine, since richer quartz than usual was broken in the second half year. For the year ending 30th June, 1890, 9586 tons were sent up from the 1800-foot level, yielding 1722 oz. 19 dwt. of gold at the mill and 278 oz. 8 dwt. (from 106 loads) at the pyrites works. The total value of the yield was £7943 5*s.* 9*d.* The average of the ore was 3 dwt. 11 gr. only, but this gave a profit of £1510 6*s.* 9*d.*, equivalent to a shilling dividend on each of the 28,000 shares. The yield was 16*s.* 9*d.* (say $4) per ton, and the cost 13*s.* 5*d.* (say $2.70) per ton. Even on this low return the company, having a paid-up capital of only £18,200, was working at a profit of 11 per cent. per annum. The New Chum Consolidated has paid £132,300 on its nominal capital of £42,000 and its paid-up capital of £18,200. The other mines will show scarcely less striking results so far as concerns the ratio of money returned to that expended.*

* As stated on pages 473 and 474, out of the 28 dividend-paying mines of 1890,

The favorable record of deep mining in this gold-field, as outlined in the above paragraphs, has led many to quote the district in proof of the richness of the deeper portions of quartz lodes. This argument ignores the unique character of the geological structure of the district. Bendigo does not furnish proof of the continuance of pay-ore to great depths in ordinary veins. If we look upon each saddle-formation as a distinct lode (which it no doubt is), we perceive that the vertical extent of the auriferous quartz is very limited. As a rule, in this district, a "reef" which can be worked to profit in three levels or stoped in two lifts may be considered above the average. The different saddles form distinct lode-formations, traversing beds of the country-rock which are different from those either above or below.

Prospecting.

It is the fact last mentioned which necessitates the most patient prospecting. Systematic development is always one of the most important features of mine-work. In such ore-deposits as these it is of paramount importance. Owing to lack of reserve-capital, and the haste made to sink the shafts deeper, there has not been that careful and patient prospecting, which the peculiar structure of the district demands, of the ground rendered accessible by each level.

Several of the richest saddle-formations have passed longitudinally through a succession of adjacent claims. Such was the main Garden Gully saddle, and such is the formation now so successfully worked from the New Chum Railway to the "180" mine. As a rule, however, the ore-shoots, particularly the richest, are small in extent, and, unless the ground is carefully cut up by drifts and cross-cuts, they are easily missed. Two instances will illustrate this. In the Old Hustlers mine, after the workings had reached a depth of over 1500 feet, there was accidentally found, last February, a good body of auriferous quartz, between the 730- and 830-foot levels, in ground which had been considered thoroughly tested. In the Hercules and Energetic mine there was found, in December, 1889, a body of quartz 8 feet square, which gave £9000 worth of gold, 46 tons yielding at the rate of 46½ ounces per ton. This was exposed in making a connection with a winze, and was as unexpected as it was valuable.

I obtained the results given by 20, and found that all save four had more than returned their paid-up capital. The average of the 20 showed an expenditure of £31,167 per mine on a nominal capital of £49,742 per mine.

At present, the work of development is confined almost entirely to the three great "lines of reef,"—the New Chum, the Garden Gully, and the Hustlers,—and more particularly to the immediate vicinity of the big mines. This is not surprising. New mines are opened as continuations of already proved properties, and there is a strong tendency to remain in the neighborhood of ground which is known to have been rich. But at Bendigo this has been overdone, and the result is that the north and south extensions of the main "lines of reef" have been but little tested, while similarly the intermediate lodes or "side-lines" have been much neglected. Since it has been demonstrated by the Geological Survey, as well as by actual mining, that the gold-belt continues for several miles on either side of the towns of Eaglehawk and Bendigo, it would be well to distribute upon shallow ground, which is as yet unworked, a little of that energy and capital which is now expended on deep sinking; for here, as in most mining districts,* the first few hundred feet are generally richer than any succeeding horizon.

As to the exploration of the "side-lines" or quartz formations which occur between the main anticlinal axes, attention has been drawn to these portions of the district by the marked success of the New Red White and Blue Consolidated, which is working a run of rich spurs on the Sheepshead lode or reef, intermediate between the Garden Gully and the New Chum. In this connection it is a curious fact that, notwithstanding the system of government assistance, no cross-cut has been driven between the main "lines of reef" in that portion of the district (in the town itself) where they have been proved to be richest. Such an exploration is recommended by the structure of the country, the richness of this section of the gold-field, and the experience of other mining districts.

Outlook for the Future.

The prospect of future deep mining on this gold-field is most encouraging. The depths yet reached are far within the capabilities

* There are exceptions, but they are not numerous. I recall the history of some of the mines on the "Mother-lode" in Amador county, Cal., where no profitable returns were obtained until several hundred feet had been sunk. As a rule, however, the increasing richness of mines with depth is a fallacy too often found among the paragraphs of a mine-report. In most districts there are few gold-mines, the first 50 to 200 feet of which did not exceed in richness any succeeding horizon.

of modern mining machinery.* There is no reason to expect any increase of water; on the contrary, the experience of deep mining in other places points to a diminution in the quantity of water as depth is gained. No chemical causes have been observed as active in this district which would tend to make the underground temperature increase other than with the usual increment of 1° for every 50 or 60 feet of additional depth. Moreover, there is every probability, from a geological standpoint, that the saddle-reef formations will be repeated, as in the 2000 feet or more already pierced; for the only cause likely to bring them to an end, the granite contact, is at a depth far beyond human reach. The cost of exploitation will increase with depth, but, as the mines are comparatively dry and the encasing rock does not become harder, it need not be expected that this increase will be material between 2000 and 3000 feet.† Below 3000 feet, the increased power and efficiency of the winding-engines, for which there is yet plenty of room at Sandhurst, will compensate for the increasing distance from which the ore will be lifted.

At present, generally speaking, 5 dwt. of gold per ton will, with a 3-foot or 4-foot reef, pay the expenses of the mine and mill. The New Chum Consolidated has shown how, at the end of the second thousand feet from the surface, 3½-dwt. ore may be made to pay. The width of the lode in the New Chum Railway, the Shenandoah, the New Chum Consolidated, the New Chum United, etc., averages about 6 to 8 feet; and it is the width of the quartz in the reefs which will determine the economical results of its extraction. In this respect there is no reason to foresee a diminution. Nor is there any apparent reason to expect a decline in the gold-contents of the quartz. In most gold-mines the first 200 or 300 feet are richer than any succeeding horizon of similar thickness; but having once passed out of the region of surface-waters there is nothing to cause one to expect a marked change. Poor zones and rich ones will alternate as hereto-

* The shafts are invariably vertical, the winding being done by steam-power with round wire-ropes and cages. Double-decked cages are not in use. Under the government regulations and the supervision of the mine-inspector, the winding-apparatus is always in good order, and the safety-appliances are frequently and regularly tested.

† The speed of sinking varies. The country traversed by the New Chum lode is less hard than that of the Garden Gully; hence, while at a depth of over 2000 feet in the "180" mine, the speed of sinking the last 200 feet averaged 20 feet per fortnight (the distance from 2440 to 2640 feet being accomplished in exactly ten fortnights), the average rate in the Victory and Pandora, on the Garden Gully line, is 12½ feet per fortnight. In both cases timbering, etc., is included.

fore; but, owing to the folding of the country in which the reefs occur and the recurrence of the beds, the mine-workings will, at greater depths, pass through portions of country already met with nearer the surface. The gold does not appear to have become more refractory with increasing distance from daylight; the percentage of pyrites has varied but little, while the fineness of the gold has not changed.*

It will be primarily a question of the economical handling of the ore, and in this respect the underground exploitation is at present as good as the treatment at the surface is bad. With mills of enlarged capacity, properly equipped with modern improvements (particularly with automatic feeders and with rock-breakers), the Bendigo gold-field should long continue to be the home of successful gold quartz-mining and the pioneer of deep sinking.

CORD-WOOD IN THE MATTING BLAST-FURNACE.

BY HERBERT LANG, MINERAL, IDAHO.

(Glen Summit Meeting, October, 1891.)

CORD-WOOD, sawn into blocks of about a foot in length, is a regular constituent of our fuel-charge at Mineral, Idaho, our work being the matting of silver-ores by fusion in a blast-furnace. The furnace is a round water-jacket furnace, of 36 inches diameter at the tuyeres, and the charge of smelting-mixture weighs 950 lbs., requiring 110 lbs. of Connellsville coke to drive it. I replace half this coke with 135 lbs. of fir-wood, cut from dead and apparently perfectly dry trees. This mixture produces as high a smelting-temperature as all coke, whence I infer that the smelting-effect of a given weight of wood is to that of the same weight of coke as 11 to 27, or 1 to $2\frac{5}{11}$. A cord of wood sawed ready for use weighs 2340 lbs., costs $5.00, and is

* In the Lazarus, at 2000 feet, I saw white quartz containing beautiful coarse gold in pieces weighing several pennyweights. The gold at the greatest depths reached is still $22\frac{3}{4}$ to 23 carats or 945 to 960 fine.

equivalent to 866 lbs. of Connellsville coke, which, at $25.00 per ton, costs $10.82, or rather more than twice as much as wood per unit of smelting-power. The saving by the use of wood plus coke, over coke alone, is therefore 75 cents per ton of ore. The principal advantage, however, is not in the saving of cost, but in the fact that a great deal of sulphur is burned off by the wood, thus allowing the use of a greater proportion of sulphide-ores in the charge, which is a point of great moment, as such ores predominate here, and we are as yet unprovided with roasting-apparatus. To offset these advantages, the wood produces a great deal more flue-dust—twice as much, I should think—and reduces the smelting-capacity about one-third. With the fuel-mixture described, I can carry only six ounces of blast; but the furnace keeps in good condition above and below, the tuyeres remain unaffected, the slag is hot and reasonably free from valuable metals, and the conditions of successful smelting are met in all respects, except as to the serious reduction of tonnage.

Mr. Dwight, in his comments upon Mr. Neill's paper on "Stone-Coal in the Lead Blast-Furnace,"* appears to infer that the coal has to be converted into coke inside the furnace before it can perform useful work. I presume he would also infer that wood has to become charcoal before it can do its smelting-work, but that such an inference is erroneous appears from the fact that our fir-wood produces but about 20 per cent. of charcoal, and that of a very poor, fragile sort. Accordingly, 135 lbs. of wood would produce only 27 lbs. of charcoal, a quantity clearly insufficient to replace 55 lbs. of coke. I therefore believe that the volatile constituents of the wood do a considerable amount of useful work in the smelting before escaping from the furnace. The smoke, which is very thick and abundant, has a peculiar nauseating odor, giving no evidence of free sulphurous acid—a circumstance which leads me to believe that the sulphur so largely burned off forms a volatile compound with the organic matters sublimed from the wood, the reaction perhaps furnishing a considerable amount of heat.

I presume that the use of denser kinds of wood, such as mountain mahogany, oak, hickory, ash, etc., would give still better results.

* Page 169 of the present volume.

RESULTS OF STREAM-MEASUREMENTS OF THE UNITED STATES GEOLOGICAL SURVEY.

BY F. H. NEWELL, WASHINGTON, D. C.

(Glen Summit Meeting, October, 1891.)

MEASUREMENTS of the amount of water flowing in many of the larger rivers of the west have been made by the United States Geological Survey, and by this means the daily discharges of these rivers have been computed for a period of over two years. These measurements have been confined mainly to that part of the country commonly known as the arid region, where the rainfall is insufficient for the needs of plants most useful to man, and the supply of water, not only for agriculture, but also for mining and all other purposes, falls below the demands made upon it. In such localities flowing water has a great value, for without it many industries are impossible.

Experience has shown that in the western part of the United States, over an area a little less than two-fifths of the whole extent of the country, there are great bodies of fertile land enjoying a genial climate, and in every way adapted to the support of a large population, except in the one particular of water-supply. By utilizing the water available, portions of this rich land can be rendered valuable. In other words, the land-values rest directly upon the prospects of a continuous artificial water-supply. The same is true of many mineral lands, which require for hydraulic mining, or for the supply of concentrating-works, stamp-mills, etc., a certain water-supply, and are impaired in value if this supply is inadequate or irregular. The pressure of this need upon the mining industry is not so widespread, or in the first instance so severe as upon agriculture; but as different regions become more thickly settled, and the two industries compete for a water-supply not sufficient for both, it is probable that the miner will suffer. But he can suspend operations with less inconvenience than the farmer. Agricultural land, if the water-supply is not continuous, becomes a desert; on the other hand, the profit of mining is greatly enhanced by the presence of agricultural

and attendant industries. Hence the subject of water-supply is both directly and indirectly interesting to mining engineers.

The arable land of the arid region being in excess of the water-supply, it is essential to ascertain as nearly as possible the extent of this supply, its fluctuations, and the causes which tend to modify it for better or worse. Recognizing this fact, Congress in 1888 passed

FIG. 1.

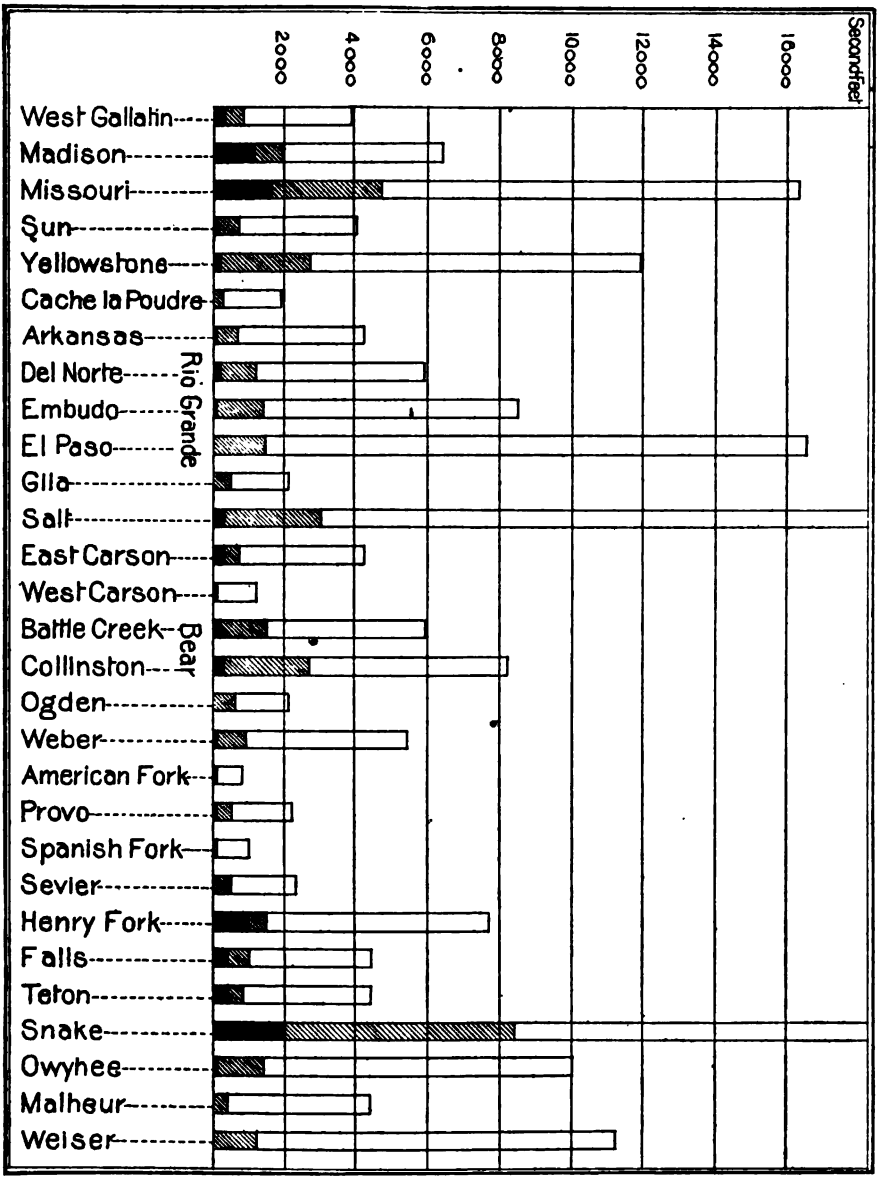
Index-Map of Drainage-Areas.

an act enabling the director of the geological survey to begin among other things the investigation of the water resources of the arid lands, and under the appropriation of October 2, 1888, field-work was immediately begun.

The index map, Fig. 1, shows by the cross-hatching the relative location of the basins within which this field-work was carried on. Although upon this scale the smaller areas are not clearly discernible,

yet the positions in general are well given. As will be seen at a glance,

FIG. 2.



Maximum, Minimum and Mean Discharges.

these drainage-areas are fairly distributed in each great hydrographic basin, there being several on the headwaters of the Missouri, the Ar-

kansas, the Rio Grande, the Gila, in the Great Basin, and on the Columbia. The greater number, however, lie along the north and south line of the high mountains which furnish so large a proportion of the waters of greatest value for irrigation in Montana, Idaho and Utah.

The point at which the river-measurements of each drainage-basin were made is to be found on the map at the point where the river leaves the cross-hatched areas. In general it is not marked in any other way, as the position would be inferred by a person examining the map since the water flows in that direction. The following brief description of these localities is given in the arbitrary order generally adopted, viz, beginning with the headwaters of the Missouri and its tributaries, and then taking in order the Arkansas, Rio Grande, and Colorado rivers, the Interior Basin, and the Columbia river.

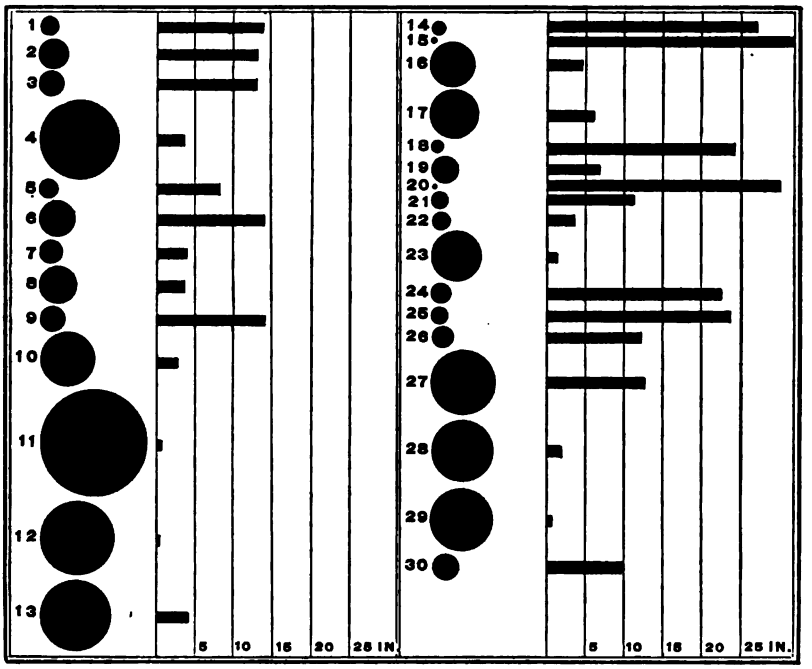
Stream-measurements in the Missouri basin were made on the West Gallatin, Madison, Redrock, the Missouri itself, and on the Sun and Yellowstone. Those on the West Gallatin river were made about 20 miles from Bozeman, Montana, above the place where water is diverted upon the Gallatin valley. On the Madison they were made in the cañon below the Madison valley, about 4 miles from the town of Red Bluff, thus giving the total amount of water flowing from the Madison valley through the cañon into the large valley above the Three Forks of the Missouri. On the Jefferson, the third of the rivers which unite to form the Missouri, continuous observations were not made upon the main stream, but a station was established at Redrock, to measure the flow of the stream of that name above the place where it enters Beaverhead valley.

Besides these upper tributaries of the Missouri, measurements were made on the Sun river about 18 miles above Augusta, and on the Missouri itself, the principal point being at Craig, about 30 miles north of Helena; also on the Yellowstone, below the National Park, near Horr, 6 miles below Cinnabar. On the map, Fig. 1, these basins can be seen, the name of the Gallatin being plainly lettered, but that of the Madison basin, to the south of this, having been omitted for lack of space. North of this comes the Missouri basin, the city of Helena being shown in its northern end. The Yellowstone basin, lying east of the Gallatin and Madison basins, is distinguishable on the map by the black spot marking the position of Yellowstone lake.

Within the catchment-area of the Platte river, a long series of observations has been maintained on the Cache-la-Poudre, one of the tributaries of the South Platte, the work for this point being for a

portion of the time maintained by the United States Geological Survey. The gauging-station is in the cañon of the Cache-la-Poudre, about 12 miles above Fort Collins. The basin is shown on the map, Fig. 1, lying to the southwest of Cheyenne. Stream-measurements

FIG. 3.



Relative Size of Drainage-Areas and Depth of Run-off.

1. West Gallatin.
2. Madison.
3. Redrock.
4. Missouri.
5. Sun.
6. Yellowstone.
7. Cache-la-Poudre.
8. Arkansas, at Cañon City.
9. Rio Grande, at Del Norte.
10. Rio Grande, at Embudo.
11. Rio Grande, at El Paso.
12. Gila.
13. Salt.
14. East Carson.
15. West Carson.
16. Bear at Battle Creek.
17. Bear at Collinston.
18. Ogden.
19. Weber.
20. American Fork.
21. Provo.
22. Spanish Fork.
23. Sevier.
24. Henry Fork.
25. Falls.
26. Teton.
27. Snake at Eagle Rock.
28. Owyhee.
29. Malheur.
30. Weiser.

have been made also on the North Platte and its tributaries by Elwood Mead, the State engineer of Wyoming, and by A. M. Van Auken, civil engineer. The drainage-area of the North Platte in Wyoming, therefore, has been cross-hatched in order to indicate that

data bearing upon the general character of the water-supply are on record.

On the headwaters of the Arkansas, gaugings have been made of the principal tributaries, the records for 1890 being quite full. The principal station is at Cañon City, and is thus above the great irrigating systems of the Arkansas valley. Measurements have also been made at Pueblo, 40 miles below Cañon City; but these have been continued for a shorter time than at the locality above.

The Rio Grande has been measured at three points, viz., at Del Norte, where it leaves the mountains and enters the San Luis valley; again at Embudo, about 130 miles below, where, having received the drainage of the San Luis valley it flows through cañons into the long, narrow valleys of New Mexico; and finally at El Paso, Texas, where the drainage from New Mexico passes through the narrow pass, below which point the river forms the boundary between Texas and Mexico. The observations at these three points give material for a study of the river as a whole, and throw light upon the regimen of streams of the southern arid countries.

The Gila, the most southern tributary of the Colorado, has been gauged in the cañons 15 miles above Florence; and the Salt River, the northern branch of the Gila, at a point about 30 miles above Phœnix. Computations of discharge on the latter stream have been made by S. A. Davidson, the engineer of the Arizona Canal Company, to whom credit is due for preserving a long series of gauge-records.

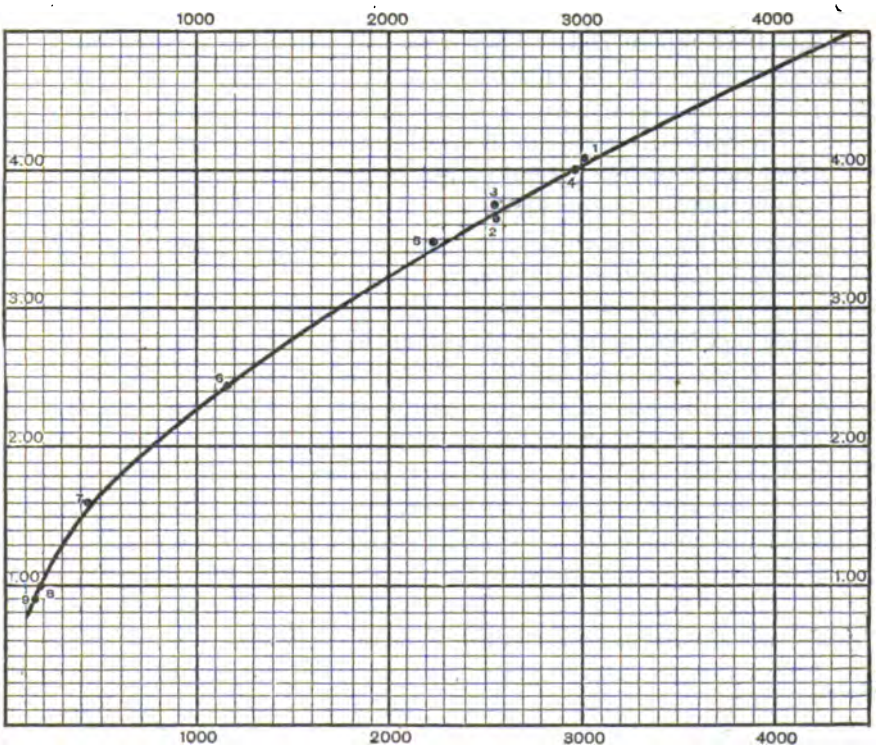
In California, the State engineering department conducted careful series of stream-measurements from 1878 to 1882, the results of which have been published in the reports of the State engineer and in the volume of *Physical Data and Statistics* issued in 1886. These reports give much information concerning the habits of the rivers which flow from the Sierra Nevada westward into the San Joaquin and Sacramento valleys.

Two rivers, the Carson and the Truckee, rising on the eastern side of the Sierra Nevada and flowing into the Great Basin, were measured during 1889 and 1890 by the United States Geological Survey. Observations have been made on the Truckee at various points, the principal of these being at Boca, California, and near Reno, Nevada. The east and west branches of the Carson have also been measured, the east branch at Rodenbah Ranch, 14 miles from Genoa, Nevada, and the west near Woodford, California, 20 miles from Genoa.

In the Great Basin, besides the observations made on the western

side, series of stream-measurements were obtained near the eastern margin in Utah and Idaho. The Bear river, the principal tributary of Great Salt Lake, was measured at Battle Creek, Idaho, at the head of Cache valley, the largest valley along its course, and again near Collinston, Utah, at the lower end of the "Gates of the Bear," thus obtaining the total outflow from Cache valley. The discharges

FIG. 4.



Rating-Curve for the Malheur River, at Vale, Oregon.

of the Ogden and Weber rivers were also measured, continuous observations being made in the cañons of each river near the city of Ogden. On the map, Fig. 1, the drainage-area of the Ogden is included within that of the Weber, since either one alone would be too small to be plainly distinguishable.

South of the drainage-basin of the Weber is that of Provo river, the principal tributary of Utah lake. The discharge of this stream has been measured in the cañon above Provo, which place is above

the heads of canals which take the water upon the fertile lands of Utah county. Between the Provo and the Weber is a very small drainage-basin, in area about one-tenth that of the Provo, and therefore too small to be shown upon a small map. The stream flowing from this area, however, the American Fork, is of considerable importance to agriculture in that part of the country, and therefore measurements have been made. The stream possesses general interest also as delivering a large amount of water from a comparatively small area. South of the Provo is the Spanish Fork basin, draining into the southern end of Utah lake. As in the case of the streams above-mentioned, the measurements on this river were made at a point in the cañon above the irrigating-ditches utilizing the waters.

The Sevier river rises in the high plateaus in the southern part of Utah and flows in a general northerly direction through cañons and narrow valleys, passing on its way through the comparatively broad Sevier valley; then entering cañons, turns toward the west and southwest, and finally loses its waters in the lake or "sink" occupying the lowest part of the Sevier desert. Measurements of this stream were made in a few instances at Joseph at the head of the Sevier valley. The longest series, however, was made below this valley, at Leamington in the cañon above the Sevier desert, at which point data were obtained as to the amount of water draining from the Sevier valley and available for use upon the desert below.

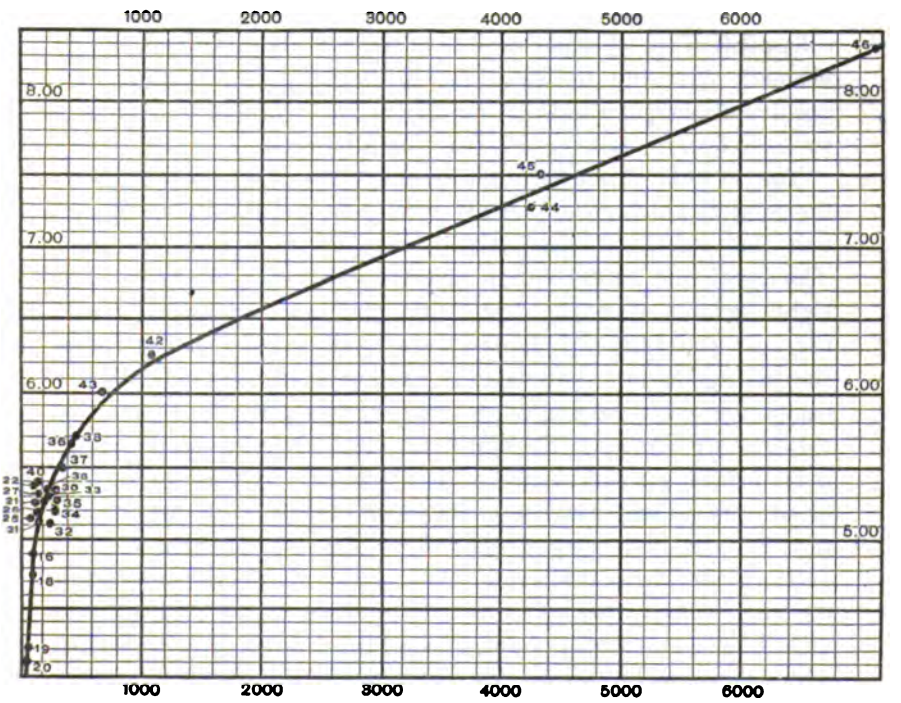
On the head-waters of the Snake, the principal tributary of the Columbia, measurements have been made, in eastern Idaho, on Henry fork above its junction with Falls river; also on the latter stream above irrigation-canals; and on the Teton, which, with the two rivers just mentioned, forms the North Fork of the Snake. On the South fork a few gaugings have been made; but the observations of river-height have not been continuous. Below the junction of the North and South forks at Eagle Rock, now known as Idaho Falls, a full series of observations was conducted during 1890, giving the discharge of the stream for that year.

Three tributaries of the lower Snake were also measured: the Weiser, in western Idaho, and the Owyhee and Malheur in eastern Oregon; gaugings of these rivers being made near the points where they enter the Snake river valley. The gauging-station on the Weiser was at the lower end of the cañon about 9 miles above the town of Weiser; that on the Owyhee, at Rigsby's ranch, 4 miles above the mouth; and that on the Malheur at Vale, Oregon.

The results of these measurements are shown in the most condensed

manner in the table of mean annual discharges (page 557). The first column at the left gives the name of the river; and in the cases of the Rio Grande and Bear, on which there are more than one station, the names of the stations also are given. In the second column is given the last month of the year for which the discharges shown in the table were obtained. For example, in the case of the West Gallatin,

FIG. 5.



Rating-Curve for the Rio Grande, at El Paso, Texas.

the year ending August, 1890, is that beginning on September 1, 1889, and ending August 31, 1890; and the next year is that beginning September 1, 1890, and ending August 31, 1891. In many instances it has not been possible to complete two full years; and hence, as in the case of the Madison, the years are not successive, but overlap. For example, the first year for the Madison is from January 1 to December 31, 1890, and the second year is from September 1, 1890, to August 31, 1891.

The third, fourth and fifth columns give respectively the maximum,

minimum and mean or average discharges for the year, in cubic feet per second or second-feet as commonly termed. The mean discharge is in all cases that for the whole year ending on the last day of the month given in the second column. The sixth column gives the total amount of water discharged during the year in acre-feet, the acre-foot being the quantity of water covering one acre to a depth of one foot, or 43,560 cubic feet. For example, the Missouri during the year furnished a quantity of water sufficient to cover over 3,000,000 acres to the depth of one foot. This unit is convenient in considerations of the efficiency or duty of water in irrigation, and is conveniently related to the second-foot in that a second-foot flowing for one day discharges a quantity for all practical purposes equivalent to two acre-feet, or, more exactly, to 1.9835 acre-feet.

The seventh column gives the drainage-area in square miles above the point at which measurements were made, and thus serves to connect the quantity discharged with the area from which it came. The eighth and ninth columns give this relation in two ways. The eighth column gives the depth which the water discharged during the year would have, if placed upon a level area of the same size as the drainage-basin. This depth is given in inches for convenience of comparison with the rainfall, which is always given in that unit by the Weather Bureau. The ninth column gives the average discharge in second-feet per square mile of area drained. It must not be supposed from the fact that these averages are computed that the assumption is made that every square mile of the drainage-area yields a perennial supply to the river. As a matter of fact the contrary is the case, namely, that while certain portions of the drainage-area yield large quantities of water, others furnish little or none—a fact which gives rise to considerable divergence in the results obtained.

The results shown in the following table are illustrated by the diagrams Figs. 2 and 3; Fig. 2 showing graphically the maximum, minimum and mean discharges in second-feet. The space from left to right indicates the discharge in cubic feet per second, lines being drawn for 2000, 4000, 6000 second-feet, etc. The solid black gives the minimum quantity discharged; the cross-hatching, including the black, shows by its length the average discharge; while the combined length of these with the open space at the right shows the maximum flood. In several instances the minimum discharge was too small to be distinguished on this scale, being less than can be shown by the width of a line, as, for instance, on the Cache-la-Poudre and Ogden, and in one case at least, that of the Rio Grande at El Paso, the river at its minimum was dry.

Mean Annual Discharges.

River.	For Year ending	Discharge in sec. ft.			Total for Year in Acre-feet.	Drainage Area. Sq. Miles.	Run-off.	
		Max.	Min.	Mean.			Depth in Inches.	Sec.-feet per Sq. Mile.
West Gallatin	Aug., '90	3,800	320	829	600,425	850	13.2	.98
" "	" '91	2,975	400	884	640,116	.....	14.2	1.04
Madison	Dec., '90	6,420	1,285	2,070	1,495,961	2,085	13.4	.99
"	Aug., '91	4,620	1,155	1,919	1,388,627	.....	12.5	.93
Red Rock	Dec., '90	675	40	148	107,249	1,330	1.5	.11
Missouri	Aug., '90	12,500	1,693	4,212	3,049,116	17,615	3.2	.24
"	" '91	16,355	1,742	5,217	3,776,452	.....	4.0	.30
Sun	Dec., '90	4,085	160	715	517,676	1,170	8.2	.61
Yellowstone	Aug., '90	11,915	510	3,034	2,196,010	2,700	15.2	1.12
"	" '91	8,975	285	2,526	1,828,239	.....	12.7	.93
Cache-le-Poudre	Dec., '89	1,960	33	283	204,868	1,060	3.6	.27
" "	Nov., '90	1,804	37	334	241,816	.....	4.3	.33
Arkansas	Dec., '88	2,760	430	860	622,640	3,060	3.8	.28
"	" '89	2,620	190	433	313,492	.....	1.9	.14
"	" '90	3,270	180	874	632,776	.....	3.9	.29
"	Aug., '91	4,230	325	1,012	732,688	.....	4.5	.33
Rio Grande. Del Norte, Col.	Sept., '90	5,930	200	1,190	861,560	1,400	13.2	.85
" "	Dec., '90	5,930	307	1,242	899,208	.....	13.7	.89
" "	Sept., '91	5,650	307	1,388	1,004,912	.....	15.4	.99
Embudo, N.M	Dec., '89	5,660	181	1,032	747,168	7,000	2.0	.15
" "	" '90	6,071	260	1,467	1,062,048	.....	2.8	.21
" "	Aug., '91	8,550	241	1,716	1,242,384	.....	3.3	.25
El Paso, Tex.	Apr., '90	4,705	0	755	546,620	30,000	.34	.025
" "	Dec., '90	7,200	40	1,327	960,748	.....	.60	.044
" "	Aug., '91	16,620	5	2,452	1,775,248	.....	1.11	.082
Gila	" '90	2,100	11	503	364,172	13,750	.50	.037
Salt	July, '89	43,489	300	2,654	1,921,496	12,260	2.9	.22
"	Dec., '89	33,794	319	2,576	1,865,024	.....	2.8	.21
"	" '90	143,288	397	3,771	2,730,204	.....	4.2	.31
"	Feb., '91	300,000	397	6,066	4,391,784	.....	6.7	.50
East Carson	Mch., '91	4,260	375	970	702,280	414	31.8	2.35
" "	Aug., '91	1,884	375	616	445,984	.....	20.2	1.49
West Carson	Mch., '91	1,284	42	206	149,144	70	40.0	2.94
" "	Aug., '91	740	42	133	96,292	.....	25.8	1.90
Bear. Battle Cr.	Dec., '90	5,980	270	1,751	1,267,724	4,500	5.3	.39
" "	Aug., '91	3,030	690	1,178	852,872	.....	3.6	.26
Collinston	June, 90	8,220	340	3,255	2,356,620	6,000	7.3	.54
"	Dec., '90	8,220	1,000	2,945	2,132,180	.....	6.6	.49
"	Aug., '91	5,000	825	1,917	1,387,908	.....	4.3	.32
Ogden	July, '90	2,178	40	616	445,984	360	23.2	1.71
"	Dec., '90	2,178	215	663	480,012	.....	25.0	1.84
Weber	Dec., '90	5,465	200	1,070	774,680	1,600	9.1	.67
"	Aug., '91	4,655	130	778	563,272	.....	6.6	.48
American Fk	July, '90	885	6	146	105,704	66	30.0	2.22
Provo	June, '90	2,260	144	532	385,168	640	11.3	.83
"	Dec., '90	2,260	200	572	414,128	.....	12.1	.89
"	Aug., '91	1,704	200	482	348,968	.....	10.2	.75
Spanish Fk	Dec., '90	1,040	50	172	124,528	670	3.5	.26
Sevier	July, '90	2,329	40	578	414,852	5,595	1.4	.10
"	Dec., '90	2,329	150	625	452,500	.....	1.5	.11
"	Aug., '91	1,386	140	544	393,856	.....	1.3	.10
Henry	Dec., '90	7,710	1,200	1,719	1,244,556	931	25.0	1.84
"	June, '91	3,180	1,280	1,458	1,055,592	.....	21.3	1.56
Falls	Mch., '91	4,440	450	1,194	864,456	594	27.2	2.01
"	Aug., '91	2,790	450	888	642,912	.....	20.2	1.49
Teton	Mch., '91	4,445	400	1,021	739,204	967	14.4	1.06
"	Aug., '91	2,360	400	704	509,696	.....	9.9	.73
Snake	June, '90	50,450	2,000	8,335	6,034,540	10,100	11.2	.83
"	Dec., '90	50,450	2,000	10,635	7,699,740	.....	14.2	1.05
Owyhee	Mch., '91	11,230	170	1,656	1,198,944	9,875	2.3	.17
"	Aug., '91	10,000	170	1,293	936,132	.....	1.8	.13
Malheur	Feb., '91	4,445	15	698	505,352	9,900	.96	.07
"	Aug., '91	2,820	15	161	116,564	.....	.22	.02
Weiser	Feb., '91	11,220	80	1,652	1,196,048	1,670	13.4	.99
"	June, '91	9,300	80	771	558,202	.....	6.3	.46

By studying the mean discharges as graphically shown on this plate, a fair conception can be obtained of the relative sizes of the

different streams. It is seen, for example, that the Snake is the largest river, being far greater than the upper Missouri, and even

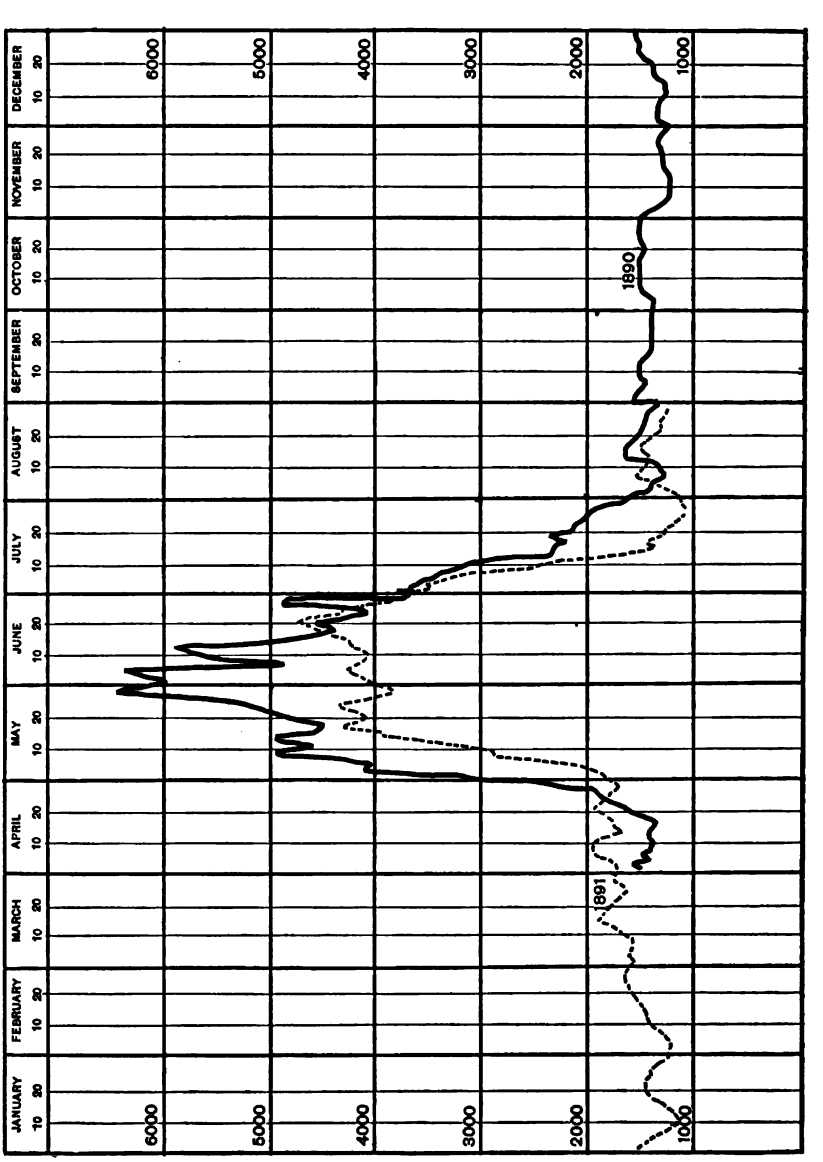


FIG. 6.—Daily Discharge of the Madison near Red Bluff, Montana, in 1890 and 1891.

greater than the Missouri and Yellowstone combined. The maximum discharges also are a matter of surprise. In the case of the

Salt and Snake rivers the maximum discharges could not be shown in their full length, since the width of the page only allows a little over 18,000 second-feet, while the great flood of the Salt river of February, 1891, reached a maximum of at least 300,000 second-feet, or nearly seventeen times the width of the diagram. The maximum flood of the Snake was, in 1890, over 50,000 second-feet, or nearly three times the width shown on the page. The character of the fluctuations of some of these streams is well shown, as for instance, the Rio Grande at its three stations. At Embudo the minimum discharge is less than at Del Norte, further up stream, and at El Paso it is still less, being zero. The average discharge, on the contrary, increases slightly from station to station in going down stream, while the maximum discharge has a still wider range, so that at El Paso the fluctuations take on a range from zero up to over 16,000 second-feet, the variations of discharge thus increasing from the headwaters down stream. The Salt river, being fed by local storms, has exhibited the greatest fluctuations during the past two or three years of all the rivers under consideration, and may be considered as typical of the streams of Arizona and New Mexico, which drain vast areas of desolate plateaus and bare mountain regions. The character of each river is to a certain extent revealed by the following table, which exhibits the relation between the average discharge and that of the greatest flood, the figures given showing the number of times that the quantity of water discharged in flood exceeds the average for the year.

West Gallatin, Bozeman, Montana,	4.4
Madison at Red Bluff, Montana,	3.2
Redrock at Redrock, Montana,	4.6
Missouri at Craig, Montana,	3.5
Sun above Augusta, Montana,	5.7
Yellowstone at Horr, Montana,	4.3
Cache-la-Poudre, Colorado,	6.4
Arkansas at Cañon City, Colorado,	5.3
Rio Grande at Del Norte, Colorado,	4.6
Rio Grande at Embudo, New Mexico,	6.1
Rio Grande at El Paso, Texas,	11.0
Gila above Florence, Arizona,	4.2
Salt above Phœnix, Arizona,	79.6
East Carson, Rodenbah, Nevada,	5.4
West Carson, Woodford, California,	7.5
Bear at Battle Creek, Idaho,	3.9
Bear at Collinston, Utah,	3.0
Ogden above Ogden, Utah,	3.4
Weber at Uintah, Utah,	5.9

Provo above Provo, Utah,	4.3
Spanish Fork above Spanish Fork, Utah,	6.1
Sevier at Leamington, Utah,	4.0
Henry Fork of Snake, Idaho,	4.9
Falls river near mouth, Idaho,	4.3
Teton near mouth, Idaho,	5.2
Snake at Eagle Rock, Idaho,	5.3
Owyhee at Vale, Oregon,	7.6
Malheur, Rigsby, Oregon	10.3
Weiser above Weiser, Idaho,	9.3

The relative sizes of the different drainage-areas are shown on Fig. 3 by the area of the black circles. The relation between these areas and the quantity of water discharged annually per unit of area is also shown by the length of the black lines to the right of the circles. The fact most prominent in this connection is the smaller amount of water discharged per square mile, the greater the size of the basin. Take, for instance, the Gallatin, Madison, and Redrock areas in comparison with the Missouri, or better, with the Rio Grande at Del Norte, Embudo, and El Paso. In this latter case the drainage-area above Embudo includes that above Del Norte, and the drainage at El Paso includes the area above Embudo, together with many small drainage areas which contribute water only in time of flood. The depth of the run-off decreases rapidly from Del Norte to Embudo, and less rapidly to El Paso. The great depth of run-off for the smaller basins is very conspicuous; for example, that of the Carson, Ogden, American fork, Henry fork, and Falls river. These small drainage-areas are made up of high, steep mountains, upon which the precipitation, mostly in the form of snow, is heavy, and from which little water escapes by evaporation, the greater part running off into the streams.

The preceding tables and diagrams give the general characteristics of many of these rivers, and show in a broad way the distribution of water in various drainage-basins. Beyond this, it is essential to know the daily habits of the more prominent streams; for, while averages enable the engineer to grasp the whole situation, they are sometimes apt to be misleading. Therefore, diagrams are given of a few of the streams, typical of those of the country in which they occur. In order to show the methods by which these diagrams were constructed, Figs. 4 and 5 are introduced (pages 553 and 555). These show the relation between the height and computed discharge in the case of two rivers, and also the observations upon which the computations are based.

Fig. 4 is an example of the ordinary rating-curve constructed for each river. The height of the river above the assumed datum

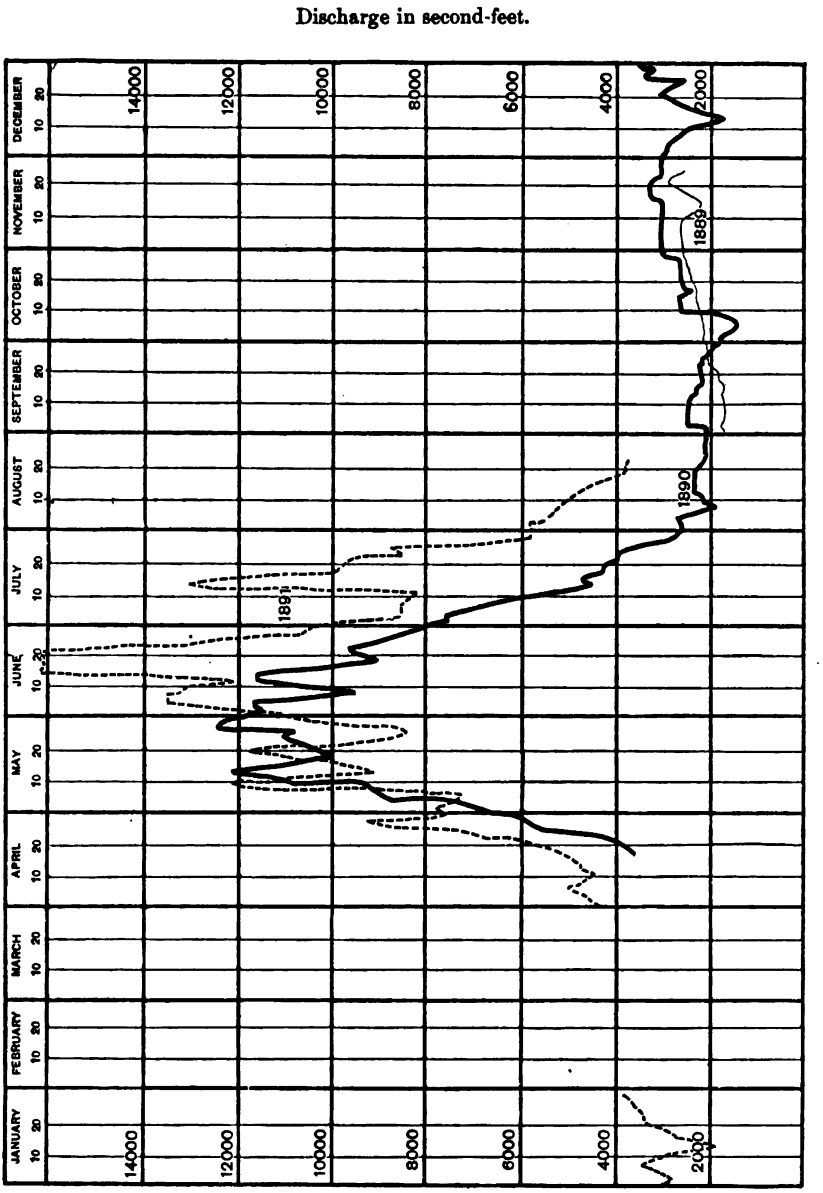


FIG. 7.—Daily Discharge of the Missouri at Craig, Montana, in 1890 and 1891.

is shown by the vertical distance above the base, while the discharge is shown on the horizontal lines from left to right, the space

between each vertical line and the next representing a discharge of 100 second-feet. The black dots with the numbers adjacent show

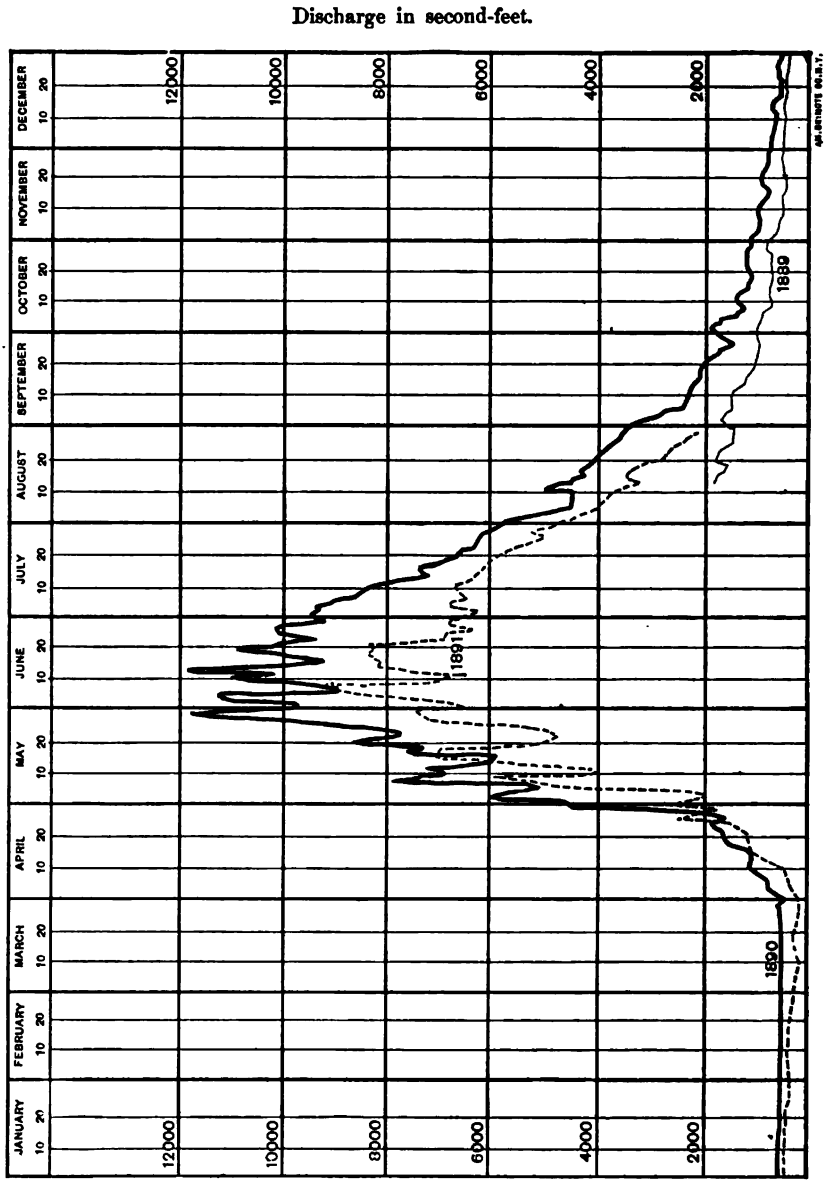


FIG. 8.—Daily Discharge of the Yellowstone at Horr, Montana, in 1889, 1890 and 1891.

graphically by their position the results of individual gaugings of the stream, the numbers giving the order in which these were made.

For example, the first gauging on the Malheur river at Vale, Oregon, gave for a height of water of 4.05 feet a discharge of 3015

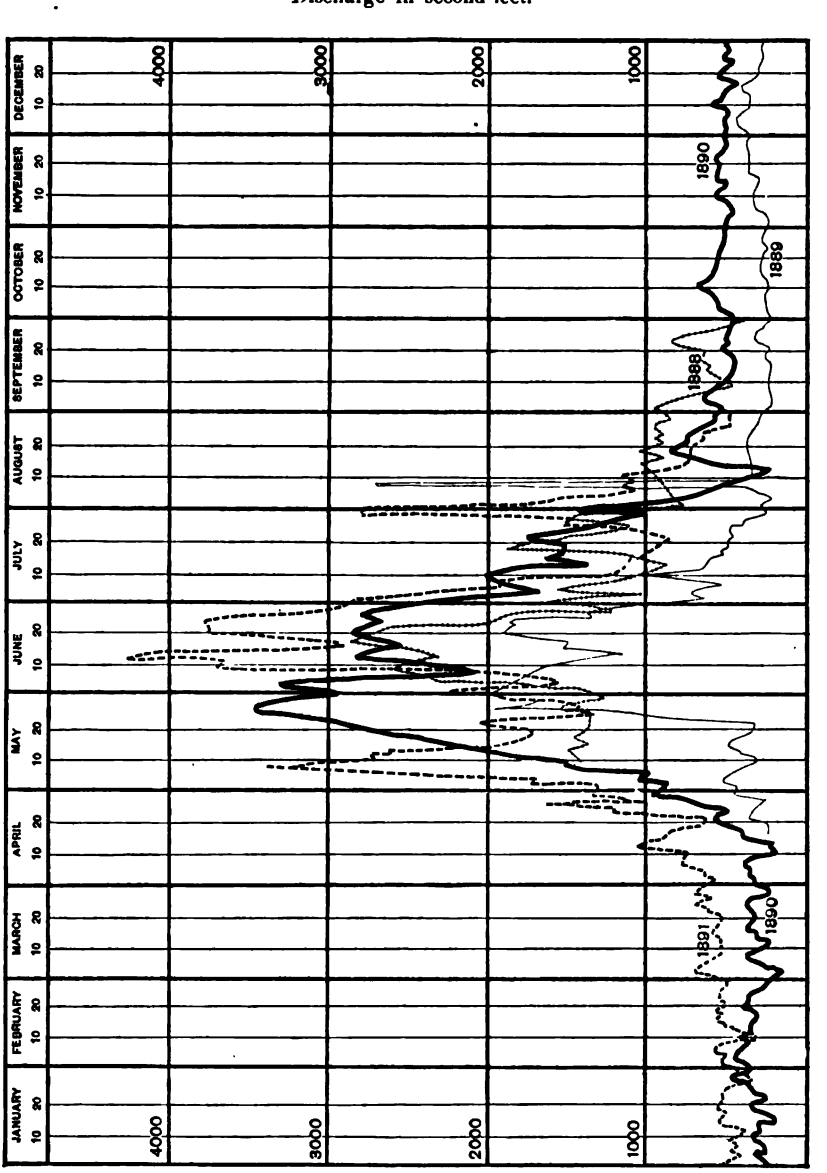


FIG. 9.—Daily Discharge of the Arkansas at Cañon City, Colorado, in 1889, 1890 and 1891.

second-feet; and thus on the plot the dot, as will be seen, is placed slightly above the four-foot line and to the right of the 3000 second-

foot line. Gauging No. 2 gave for a height of water of 3.70 feet a discharge of 2568 second-feet, and No. 3 for a height of 3.75 feet a discharge of 2561 second-feet. Nine gaugings are shown by dots on the sheet in their proper locations. Through these points a curve has been drawn in such a position that they lie at equal distances on both sides of the line if not cut by it.

This line is assumed to give the relation between the height of the water and the discharge, and from its position on the diagram a table is constructed giving for each tenth of a foot rise or fall the corresponding amount of water flowing in the stream. For example, the point at which the curve cuts the line marking the river-height of two feet corresponds to a discharge of 785 second-feet; at 2.10 gauge height to a discharge of 835 second-feet; at 2.20 to a discharge of 925 second-feet, and so on. This rating curve and corresponding table when once constructed serve to give at a glance the flow of the stream when the gauge-height is known.

On the banks of each of the streams measured by the Geological Survey, at the selected gauging-station, is fastened a stout gauge, marked with feet and tenths of a foot, and by this the height of the water can be readily ascertained. This gauge, in general slope, coincides with that of the bank of the river, so that it will not offer any obstruction to the stream or be exposed to destruction by drift-wood. The marks are placed upon it at distances corresponding always to vertical feet and tenths above some arbitrary but permanent bench-mark. The observer at the station notes the height of the water on the gauge once, or in some cases twice, a day, and sends a report weekly to the office at Washington. Here the observations of river-height are, by means of the rating-table, converted into discharges, and are recorded and plotted upon diagrams.

The rating-curve for the Rio Grande at El Paso, Tex., given in Fig. 5, shows a curious form of curve, and also the number of observations taken at comparatively low stages, and thus exhibits the range in results obtained under varying circumstances. In many instances the wind, blowing either up or down through the narrow pass, decreased or increased the discharge corresponding to a given gauge-height. Other factors also enter to give, in many cases, a considerable range of discharge for any given height. In the work of the Survey, however, it has been found impossible to discriminate between the discharges for various directions and velocities of wind or for rising and falling water; and the attempt is made to construct the rating-curve of such a character that it averages all these minor

conditions, and thus, while in any one instance the discharge as taken from the rating-curve for a given height may not be as accurate as a

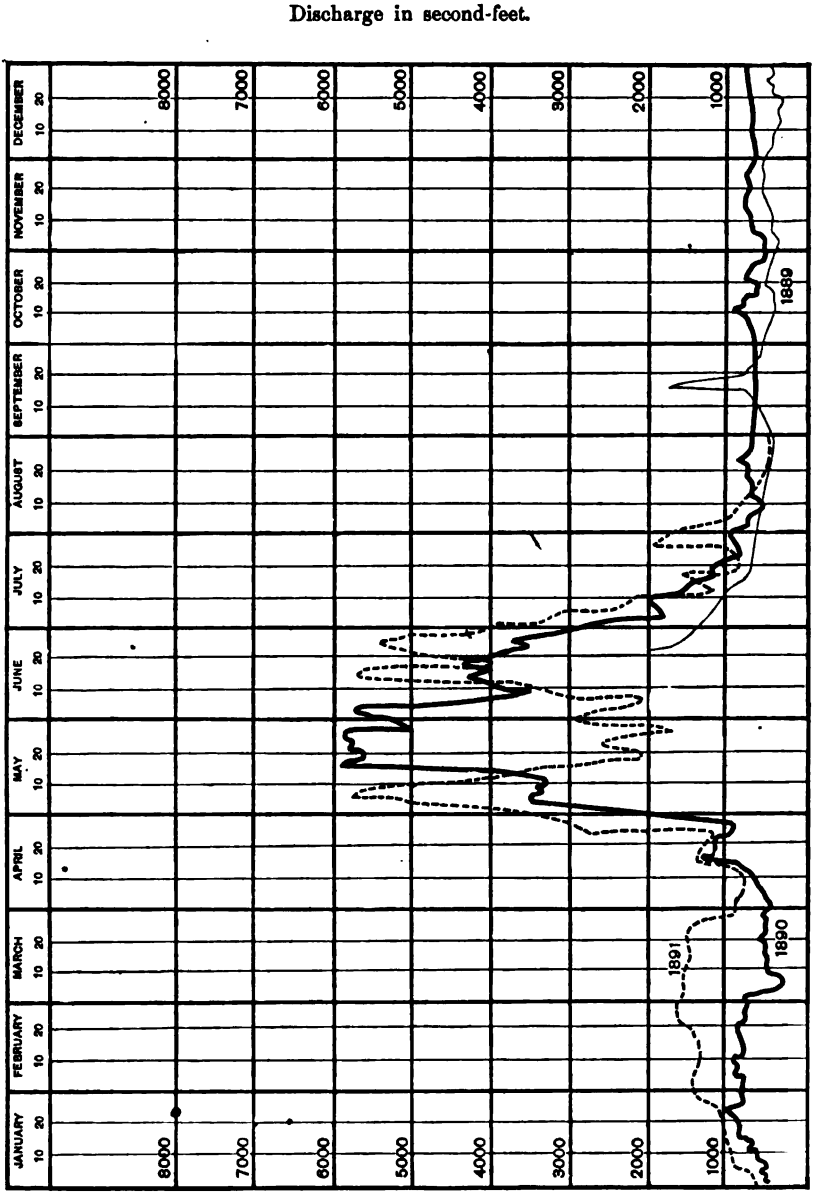


FIG. 10.—Daily Discharge of the Rio Grande at Del Norte, Colorado, in 1889, 1890 and 1891.

measurement made on that date, yet in the long run, through months

and seasons, the averages should possess a considerable degree of accuracy.

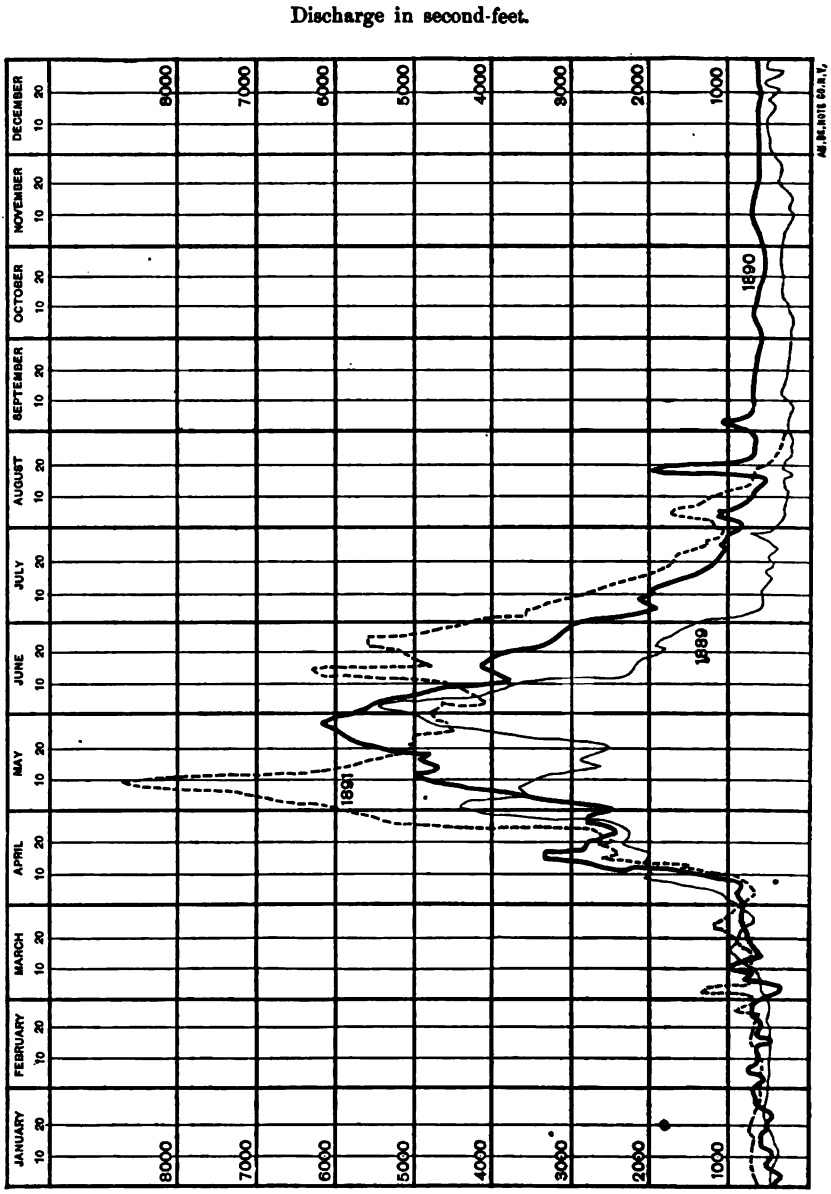


FIG. 11.—Daily Discharge of the Rio Grande at Embudo, New Mexico, in 1889, 1890 and 1891.

Figs. 6 to 16 show the daily discharge of various rivers, principally for the years 1889, 1890, and 1891. On these diagrams the

space from left to right indicates the time progressively through the year, as shown by the headings giving the names of the months. As

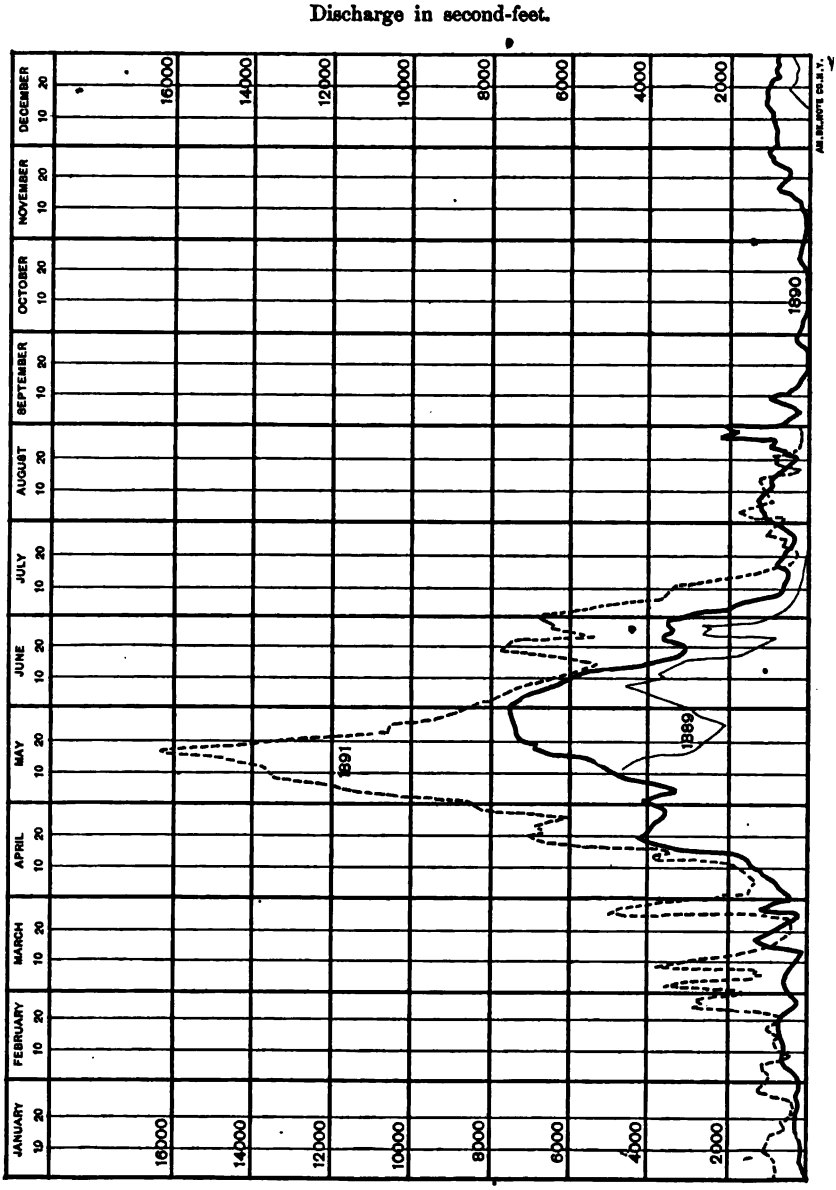


FIG. 12.—Daily Discharge of the Rio Grande at El Paso, Texas, in 1889, 1890 and 1891.

it would be impossible on such small diagrams to show each day, only sufficient lines are given to serve as a guide, these being for the

tenth and twentieth days of each month, and the last day or beginning of the next month. The fluctuations of the rivers on intermediate days can be ascertained by dividing the spaces proportionally by the eye. The distance from the base upward in these diagrams represents the discharge of the river in second-feet. In most of them, for clearness, the horizontal lines have been drawn only for thousands of second-feet, but the intermediate values, as in the case of single days, can be obtained by proportional subdivision. The full black line represents, in all cases, the discharge for 1890, and the dotted line that for 1891.

Fig. 6 gives the discharge of the Madison river near Red Bluff, Madison county, Mont., from April 4, 1890, to the end of August, 1891. On April 4th, the discharge was 1520 second-feet, and therefore the discharge-curve begins at a short distance to the right of the vertical line which separates March from April, and a little over half-way between the horizontal lines for 1000 and 2000 second-feet. On April 5th, the discharge was 1580 second-feet, and, as shown by the diagram, the line of discharge rises a trifle. On the sixth, the discharge was 1420 feet, and therefore the discharge line descends. The fluctuations, then, for a week or more, are very small, scarcely distinguishable, and on April 17th the lowest water for the month, 1380 second-feet, was reached. From that time the advance was constant, and by the 1st of May the discharge was over 3000 second-feet. The river still continued to rise with minor fluctuations, such as that of May 11th and May 18th, probably due to changes of temperature, a cold day checking the flow. The maximum of 6420 second-feet was reached on the 29th of May, followed by a slight decline and return on the 4th of June, from which time the river continued to fall, with various fluctuations, through June and July, reaching a minimum in the early part of August. Then, as shown by the diagram, the fluctuations continued through the rest of the year, generally below 1500 second-feet. After the 31st of December the discharge-line is continued, but now as a dotted line on the left side of the diagram, the fluctuations during January and February being similar to those of November and December of the year preceding. During May and June of 1891 the discharge, as shown by the position of the dotted line below the solid line, was far less than that of the preceding year, the maximum, reached on the 20th of June, being only 4620 second feet.

Fig. 7 gives the discharge of the Missouri at Craig, Mont., about 30 miles north of Helena and 50 miles above Great Falls. The

thin continuous line in the lower left-hand corner gives the discharge of the river at Cañon Ferry, about 15 miles eastward from Helena,

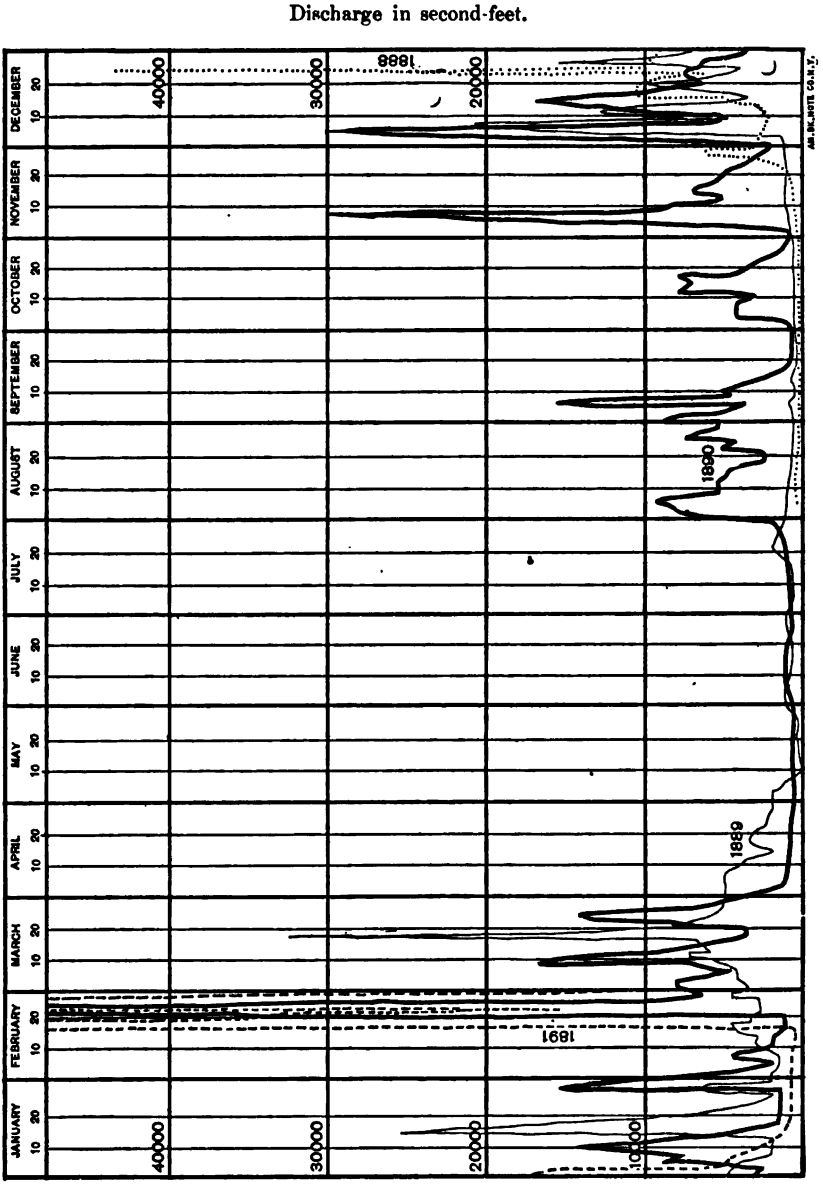


FIG. 13.—Daily Discharge of Salt River, Arizona, below its junction with the Verde, in 1889, 1890 and 1891.

for the fall of 1889. It is to be noted on this diagram that the discharge for 1891 exceeded that of 1890, although the contrary

was true in the case of the Madison, one of its principal tributaries, thus indicating that the greater part of the flood came from some other source.

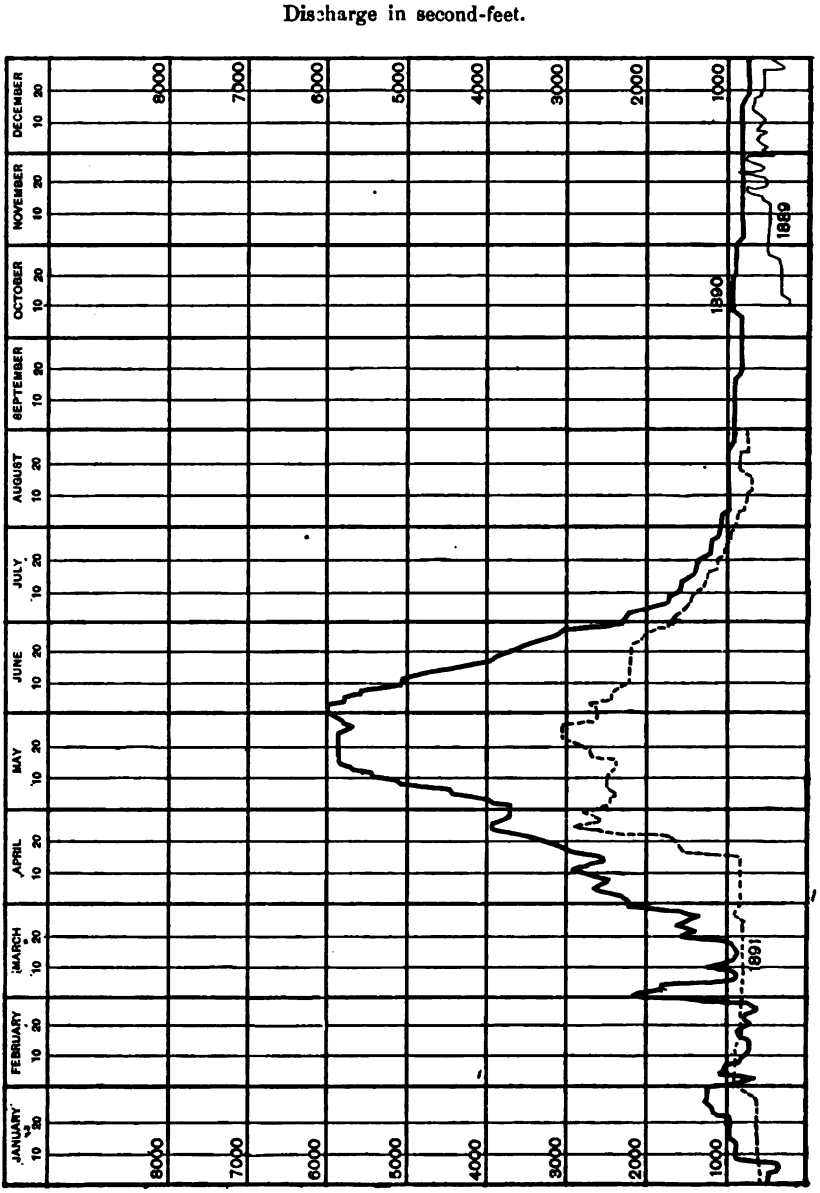


FIG. 14.—Daily Discharge of Bear River at Battle Creek, Idaho, in 1889, 1890 and 1891.

The discharge of the Yellowstone at Horr, Park county, Mont.,

is shown on Fig. 8 from August 12, 1889, to the end of August, 1891. Here, as in the case of the Madison, the discharge for 1891

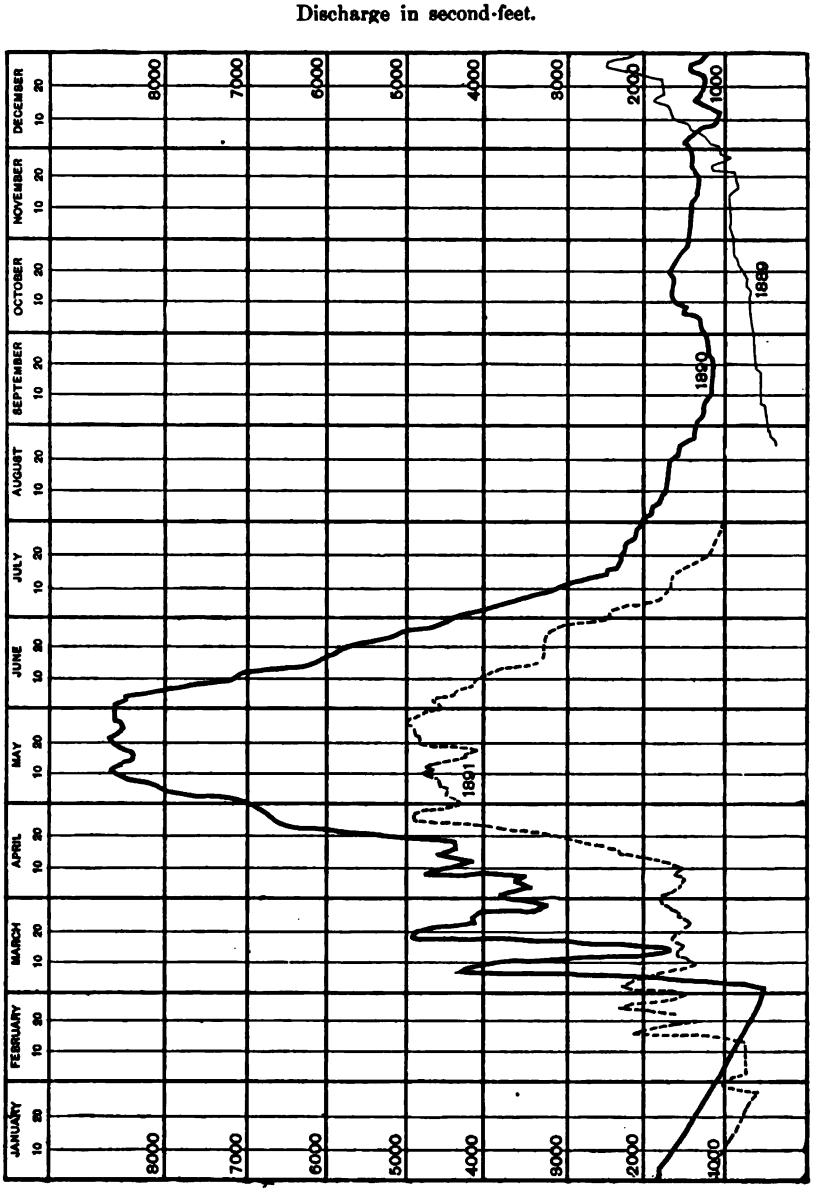


FIG. 15.—Daily Discharge of Bear River at Collinston, Utah, in 1889, 1890 and 1891.

was decidedly less than that for 1890. One feature characteristic of these northern rivers is well shown on this diagram, namely, the

rapid fluctuations of discharge due to changes of temperature in the mountains. The river-water comes largely from melting snow, and as a consequence any change in the weather, though comparatively slight, shows itself upon the flow of the streams.

Fig. 9 gives the discharge of the Arkansas at Cañon City, Colorado, from May 10, 1888, to the end of August, 1891, and, although somewhat confusing, is of especial interest as showing the fluctuations for nearly four years. The discharge for 1888 is shown by a thin line marked by dots, and, as can be seen by careful inspection, in general follows the discharge for 1890 quite closely. For 1888, however, the discharge line is very low, this being a year of unusual drought. During June it barely reaches the discharge of 2000 second-feet. The maximum for the year is that of August 9th, when, owing to a storm of unusual violence, a sudden and destructive flood occurred, tearing out many bridges and canal-headworks along the stream. Floods of this character, which rise in a day or even in a few hours, are far more to be dreaded than the gradual increase during spring. The year 1891 is characterized by a flood on June 13th, reaching a maximum of 4230 second-feet, after which the river declined rapidly. Another noticeable feature is the low water during the latter part of May, 1891, at a season when, in 1890, the water was high.

Figs. 10, 11, and 12, showing the discharge of the Rio Grande at Del Norte, Embudo, and El Paso, are introduced in order to exhibit the behavior of the same river at three different points, viz.: at its head-waters and in its lower portions. The diagram for El Paso, Fig. 12, is in vertical scale double that of the other two, in order to exhibit the extent of the flood of 1891. In other respects the figures are comparable. Examining these in succession, it will be seen that during low water in 1889 there was a decrease in amount from the upper to the lower stations, the river at El Paso being dry. During 1890 the discharges at three points were similar, the early floods of April, as might be expected, being greater on the lower reaches of the river. In 1891 the most conspicuous feature is the flood of early May, the maximum in second-feet being for Del Norte 5650, on May 7th; for Embudo, 8550, on May 10th; and the top of the flood reaching El Paso May 18th, with a maximum of 16,620 second-feet. As will be seen from the great increase in the lower river, the greater part of this flood must have come from places below Embudo. This emphasizes the statement before made that on the lower courses of these southern rivers the range of fluc-

tuations increases enormously, although the average discharge for the whole year may be but slightly above that for the upper part of the river.

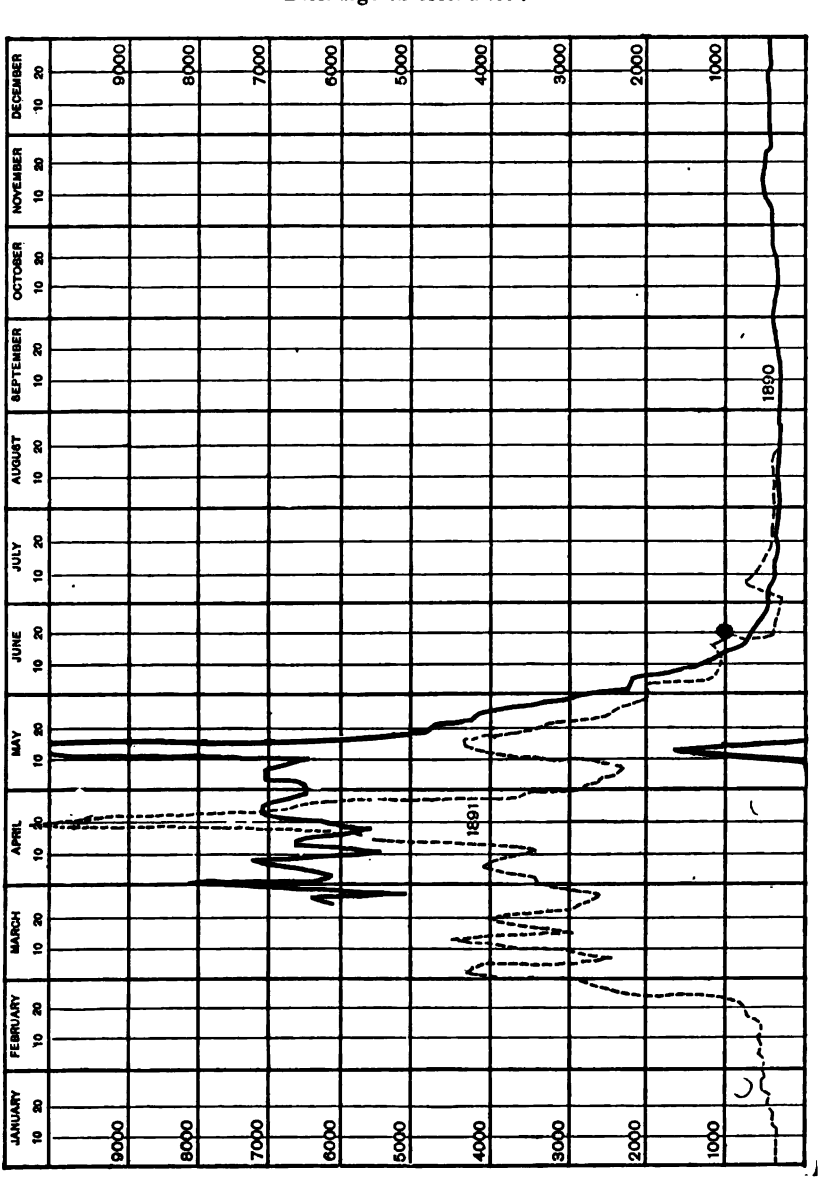


FIG. 16.—Daily Discharge of the Owyhee River at Vale, Oregon, in 1890 and 1891.

Fig. 13 shows the discharge for Salt river, Arizona, at a point

below its junction with the Verde, the records of discharge, as previously stated, being kept by the engineer of the Arizona Canal Company. Comparing this with the diagrams previously shown, the greatest possible contrast is noticeable. In this case there are no spring floods. During May and June the river is at its minimum; and finally, when the water does rise, it is by short, sudden impulses, rising and falling within a day or two, the decline being almost as sharp as the increase. The most remarkable features of this diagram are the great floods of February. That of February 22, 1890, reached a maximum of over 143,000 second-feet, or nearly three times the height of the diagram, while that of February 24, 1891, was more than twice as great as the first, reaching a maximum, probably of 300,000 second-feet, or over six times the height of the diagram. This great flood destroyed a part of the Arizona Company's irrigating-canal, and, passing down the valley to the Gila, washed out bridges and headworks of irrigating-ditches along its course, and finally, escaping into the Colorado, caused that stream to open an abandoned channel leading into the depression of southwestern California.

Figs. 14 and 15 exhibit a type of river in great contrast to that given above. These diagrams, giving the discharge of the Bear river at Battle Creek, Idaho, and at Collinston, Utah (the points previously described), show the extraordinarily low water of the fall of 1889, and also exhibit clearly the diminished discharge for 1891. In general, the discharges at the two points follow each other closely, that at Collinston being in all cases greater on account of the added influx from Cache valley, Utah.

Fig. 16, showing the discharge of the Owyhee river of southeastern Oregon, is typical of streams which receive the greater portion of their water from comparatively low mountain masses. Here the discharge occurs earlier in the season than in the case of the Bear, the maximum being in April or early May instead of in late May and June. The great fluctuations of volume, due to changes of weather, are also seen in the irregularities of the diagram. It is noticeable also that the discharge for 1891 was less than that for 1890, as was the case on most of the rivers measured by the Geological Survey. One flood, that of May 12, 1890, reaching a maximum of 11,230 second-feet, was too great to be shown on the diagram to its full height.

Discussion.

Remarks were made by Messrs. Birkinbine, Coxe, Platt, Fernow and Raymond in general discussion of the above paper—mainly in recognition of the importance of the subject to mining as well as agriculture. In answer to questions, Mr. Newell said:

Observations of rainfall are made by a considerable number of observers, scattered over the country; but the measured rainfall does not correspond directly with the volume of the streams. Many conditions influence this relation, particularly the circumstance that the rivers under examination receive their principal water-supply from melting snows in the mountains.

With regard to the amount of water which flows out of sight, beneath the surface, in some of the western rivers, it should be borne in mind that the "immense" outflows derived from artesian wells are really very small in comparison with a river. An artesian well which flows five cubic feet per second is a very large one; but five cubic feet per second is insignificant for a river.

THE MAGNETIC CONCENTRATION OF IRON-ORE.

(A Discussion held at the Glen Summit Meeting, October, 1891, together with Communications subsequently made to the Secretary.)

[Note.—The paper of Mr. Hoffman on "Practical Results in the Magnetic Concentration of Iron-Ore" (page 602); that of Mr. Langdon on "The Use of Magnetic Concentrates in the Port Henry Blast-Furnaces" (page 599); and those of Mr. Landis on "The Determination of Iron in the Tails from Magnetic Concentration" (page 609) and "Note on Sampling Iron-Ore" (page 611), were all prepared as contributions to this discussion.]

The papers of Mr. Langdon and Mr. Hoffman were presented.

W. H. Hoffman, Croton Mines, N. Y.: I will add a few remarks, by way of answer to inquiries for further details. The Croton Mines are particularly well located for cheap mining. The ore-bed is very large, and so lies in relation to the railway tracks of the Harlem and other railroads that a very large proportion of our mining for several years to come can be done above these tracks. The concentrating-mill is now 400 feet from where we are working in the quarry. In blasting we never remove less than 2500 tons at a blast,

and sometimes 4000 tons. We haul off and break the large pieces immediately afterwards. With these ores we mix from one-quarter to one-half of waste from the old mining operations which have been going on for six or seven years, and in which about 50,000 tons of foundry-ore were shipped. The dump is immediately alongside of the mill. All the ore from the dump or the quarry has to be lifted to the crushers, a height of 21 feet.

The ore is crushed in the initial crusher to pass through a 2¼-inch sieve. It then goes to the top of the roasters, nearly all of which are so arranged as to be automatically drawn, and require no help at the bottom. The ores are conveyed automatically into the Sturtevant mill and thence to the screen, and the material rejected from the screen comes back to the mill. We first reconstructed from a former mill and of course retained much of the old machinery. But we have purchased new machinery in order to experiment with the crushers and rolls, to see which would be the best adapted when we came to the point of mining, as we propose to do, from 1500 to 2000 tons in twenty-four hours. Hence, in these experiments, we have made very careful notes of our wear and tear and of the power required for the machine. It was evident that the multiplicity of machinery would have to be dispensed with, in order to make magnetic separation really profitable. The Sturtevant mill was one of our experiments, and we have now run it for seventeen months. The machine requires 94 horse-power to make 880 revolutions per minute, and to reduce say 20 tons of rock per hour, so that it will pass through a 12-mesh screen. Strictly speaking, it passes 22 tons per hour, but 20 per cent. of that comes back to the mill, having been rejected by the 12-mesh screen. For screening we have tried several kinds of wire-cloth and several kinds of plates. We first used Russia iron and now we use steel plates, rolled hard and then punched with slots one-twelfth of an inch wide, three-eighths of an inch long and three-tenths of an inch apart, so as to give strength enough to the plates. These wear admirably. We have just taken out a set of screen-plates which have been in use for thirteen months. We consider that an extraordinary wear. By means of complete arrangements for automatic handling at all stages of our operations, including the final loading of the railroad-cars from our storage-bins, we believe we are getting economy down to a point never reached before.

The crude ore, concentrates and tailings are analyzed every day. Last year our tailings carried from 10 to 14 per cent. of iron; they

are now running from 7 to 8 per cent. We have devoted a great deal of attention to that one point, because in that lies, of course, a large amount of the profit in the business. We assume our ore to average 38 per cent. of iron, though it really carries more. In July and August it ran nearly 39 per cent. At the same time we should not think of running a large mill where we would have more than 7 or 8 per cent. left in tailings.

As to the low cost of our mining—a point which has attracted considerable attention—I would say that we sent to several parties, some time ago, a preliminary bid, carefully prepared under the advice of Mr. Birkinbine; and the bids we received were in all cases below the figures named in my paper—viz., $1.38 per cubic yard. That shows what reputable contractors think of the mine as a place to work in for profit. These were all men of experience in mining iron-ore, as well as quarrying, and all of them examined the property in person or through their own engineers; so that we know that the price at which we have contracted to-day is higher than what we can get it done for on still larger amounts. We are mining now from 580 to 620 tons in twenty hours. As to the use of the concentrates I cannot speak from personal knowledge, not being a furnace-man; but we have had many letters from consumers expressing satisfaction.

The two papers of Mr. Landis were then read by the Secretary.

THE PRESIDENT presented, as pertinent to this discussion, the following letter from Mr. Woodbridge, dated September 30, 1891, and giving the results of a series of determinations made at his request:

T. R. WOODBRIDGE, Port Henry, N. Y. (Communication to the President): In a letter to you on May 9th last, I gave a table of seventy-one determinations of iron in the Old Bed tailings from several magnetic separators, showing the percentages of the total iron contents which was insoluble in dilute hydrochloric acid. As you remember, the letter was a continuation of a discussion on the claim made by representatives of certain separators, that nearly all of the iron in the tailings was "non-magnetic;" and, therefore, the comparatively high percentage of iron in the tailings were not an indication of inefficient work of the machines. The averages from that table were as follows (see page 578):

The amount of iron dissolved by hydrochloric acid was then supposed to indicate approximately the amount of magnetite in the mass; that is, it was assumed that, of the iron-bearing minerals,

little else than magnetite gave up iron to dilute hydrochloric acid. At that time, however, and in former discussions, I said that this method gives but a rough approximation; since, although *all* the magnetite does dissolve in the acid, the amount of iron given up by the silicates depends on the strength and temperature of the acid used and on the time it is allowed to act upon the powder.

Averages from Table of Irons.

No. of Samples taken.	Machine.	Total Iron in Tailings. Per cent.	Insoluble in HCl. Per cent.	Soluble in HCl. Per cent.	Per ct. of total which was Insoluble.
17	No. 1	9.01	1.90	7.11	21.1
21	No. 2	9.97	1.96	8.01	19.7
38	Averages,	9.36	1.92	7.44	20.5

Later investigations, although not entirely satisfactory, have developed some facts of interest. For instance, Rosenbusch says that "all iron bearing minerals may be separated from non-ferruginous minerals by means of an electro-magnet." In experimenting with Old Bed tailings, I have found that three passes under the poles of an electro-magnet left a residue consisting mostly of apatite, quartz and feldspar, and containing from 0.30 to 0.60 per cent. of iron. A few more passes left a perfectly white residue with but a trace of iron, and I have been unable so far to find in the Old Bed ores any compound of iron which the electro-magnet refused. Even crystals of apatite containing less than 1 per cent. of iron were easily removed from the powder without being brought into contact with the poles of the magnet. It seems to me that this effectually disposes of the term "non-magnetic" iron, so far as Old Bed ores and their associated minerals are concerned. In this connection it may interest you to know that I have found pieces of Michigan hematite to be strongly attracted by an electro-magnet run by two one-half gallon Bunsen cells. The pieces weighed about one-third of a gramme, contained less than 0.20 per cent. of FeO, and jumped one-eighth of an inch to the magnet. I would not, however, advise the investment of much capital in the magnetic concentration of hematite ores on the strength of this experiment.

The problem as to the amount of magnetite in the tails I have found great difficulty in solving, and have as yet discovered no satisfactory method of determining the mineral. A weak bar-magnet

would seem to offer a ready means, but owing to inclusions (even with the ore crushed to 100-mesh) I have been unable to obtain magnetite purer than 69.60 per cent. of iron; and the residue even then contained magnetite. Neither are chemical methods entirely satisfactory.

The tails consist principally of pyroxene, amphibole, biotite, feldspar, apatite, quartz and magnetite. An experiment showed that the first three of these minerals gave up to hydrochloric acid (1.07 sp. gr.) from 25 to 95 per cent. of their total iron contents. As they contain as high as 15 per cent. of iron, the table given above shows too low a percentage for the iron in the silicates and too high a percentage for the iron in the magnetite. Hydrochloric acid of lower specific gravity gave similar results.

Dilute sulphuric acid dissolved little more than the apatite and carbonate, and the strong acid decomposed a portion only of the silicates and appreciably attacked the magnetite. Treated with cold dilute hydrofluoric acid for four hours, over 98 per cent. of a weighed portion of magnetite passes into solution, although this acid is supposed to have no action on magnetite.

Results with other acids having been so far largely negative, I will not detail them here. I hope soon to determine accurately the amount of magnetite lost in the tailings; but I think at present that, for practical purposes, two-thirds of the total iron contents represented by the above table may be considered to be due to magnetite. That is, that the average tails represented contain about 8.62 per cent. of their weight of magnetite.

R. W. RAYMOND, New York City: I am reminded by Mr. Woodbridge's letter of a paper contributed by Prof. Egleston to our *Transactions* many years ago on the analysis of rocks (St. Louis Meeting, 1874, *Transactions* iii., 94), in which he described a process of combined chemical and mechanical analysis invented by Mr. Fouqué of the College of France. The first stage was simply a reduction to powder. Of the second stage, Prof. Egleston says:

"In order then to separate those parts which contain iron from those which do not, the powder is submitted to the action of an electro-magnet, worked by eight or nine ordinary Bunsen cells. The mineralogical elements of the rock are thus divided into two parts; those which contain iron, even in very small proportions, whether combined with silica or not, and those which contain no iron."

The paper goes on to describe in detail the method of using the

magnet. I remember very well that Dr. Drown, then Secretary of the Institute, was surprised at the statement about the silicates. He consulted me and others, and we thought Prof. Egleston must have omitted some intermediate step of treatment, such as exposure to a stream of hydrogen. But when we appealed to him he adhered positively to his first statement, and so it went into the *Transactions* on his authority. Subsequent inquiry abroad resulted in a confirmation of his paper, and we gave up what we had fancied would prove a good joke on him! I suppose it is now generally believed that the magnetic property in magnetite is due to its protoxide; in other words, that the protoxide of iron is magnetic. As the silicates all carry their iron as protoxide, the inference is that iron in such combinations is still more or less susceptible to the magnet. I confess that what Mr. Woodbridge says about hematites is much more surprising to me. I fancy that in such a case there must be some admixture of protoxide.

C. B. DUDLEY, Altoona, Pa.: We have recently been examining in our laboratory some paints which are oxides of iron, and to our surprise when we stick an ordinary hand-magnet into them we can pull out almost any quantity of the mass. Most of them have been roasted; we have not made a definite test as to whether there is protoxide there, but the amount of material which adheres to the magnet is startling.

I would ask whether the method of separation of the associated minerals in these concentrates or tailings by means of the Thoulet solution has been tried. I am not familiar enough with the differences in specific gravity to know whether it would work, but it occurs to me that possibly a heavy solution might be made which would allow the magnet possibly to separate the lighter minerals.

JAMES W. NEILL, Salt Lake City, Utah (Communication to the Secretary): With reference to the theory that FeO is magnetic, and that the magnetic character of iron-bearing minerals is due to it, I would mention some facts which do not support that view. The slags of the lead blast-furnace are protosilicates of iron, including also as bases (as do many of the natural silicates) lime, magnesia, alumina, etc. These slags carry from 28 to 52 per cent. of FeO, according to the technical and commercial conditions of smelting. As made to-day in Colorado, they vary from 28 to 34 per cent. in FeO, silica ranging from 30 to 35 per cent., and lime (including the "lime bases") from 16 to 28 per cent. According to the above theory, these slags should be attracted by the magnet. But this is

not the case with those of Colorado last named. With higher percentages of FeO, they become susceptible to the magnet, and the slag having 52 per cent. of FeO (the "quarter-slag" of old times in Leadville) is almost completely so. Just where the line is to be drawn, I have not determined; but I know that samples carrying 40 per cent. of FeO were not attracted. I made this observation in the course of experiments undertaken with the purpose of extracting from pulverized silver-lead slags the enclosed "shots" of argentiferous matte. The matte was found in all the experiments to be magnetic, and I was able to extract with the magnet from crushed "shells" (passing an 80-mesh screen) from 33 to 65 per cent. of the total silver contained in the sample.

MR. RAYMOND: This interesting statement does not seem to me entirely conclusive, without more definite quantitative details as to the intensity of the magnet employed. The fact that a simple increase of the percentage of ferrous oxide does at some point develop the visible effects of the magnetic attraction, suggests at least that the attraction exists at lower percentages, only not in sufficient strength to overcome the inertia of the total compound. Its energy might be augmented either by increasing the proportion of ferrous oxide, or by increasing the intensity of the magnetic force.

WILLIAM KENT, New York City: It seems to me that a partial analysis might be made, based on the difference of specific gravity between the magnetite and other iron-bearing minerals. If we should crush the material to a very fine powder, and then blow it with a gentle current of air, the heavier particles would settle nearest the current and the lighter particles farther away. This is the way emery is treated in the manufacture of emery powder. Or it might be done by water-separation, as in separating graphite, by carrying it through several tubs of water placed in a row, one after another. These would separate the minerals in the order of their specific gravity, and the microscope could then be used to determine the nature of the different minerals.

FRANK FIRMSTONE, Easton, Pa.: I have never experimented personally with the Thoulet solution, but something was done with it two years ago in the Pennsylvania Geological Survey, and according to what I heard of it at that time I should think the Thoulet solution would be better than any other method for practical purposes. The separation is exceedingly neat and almost absolutely accurate, provided the fineness is not pushed to too great a point. When it reaches too great a point the material will not settle with sufficient

rapidity through the solution; but my impression is that it would not be necessary to crush it to that point in order to get all necessary practical information as to the amount of magnetite in a given ore.

This solution is iodide of mercury and iodide of potassium (sometimes iodide of barium, as an improvement on the iodide of potassium). It is perhaps doubtful whether it will satisfactorily float the heavier varieties of hornblende. The specific gravity of the Thoulet solution is above 3. A still heavier solution is that of the boro-tungstate of cadmium.

E. K. LANDIS, Pottstown, Pa. (Communication to the Secretary): The use of Thoulet's solution is scarcely practicable, because the magnetite is not free, but mixed with particles of other minerals. In many microscopic examinations of tailings, I have never found pure magnetite when the machines were working well. In the Croton tailings I have often found extremely small crystals of quartz to be attracted by the magnet, although no magnetite could be seen with the naked eye; but on magnifying to 100 diameters a small magnetite crystal was always discovered completely imbedded in the quartz crystal.

T. M. DROWN, Boston, Mass. (Communication to the Secretary): Thoulet's solution, first proposed by E. Sonstadt, but generally known through the researches of Thoulet, is a solution of potassium-mercuric iodide, the maximum density of which, according to von Goldschmidt, is 3.196. The cadmium boro-tungstate solution, mentioned by Mr. Firmstone, is that proposed by Dr. Klein. It has a maximum specific gravity, at 15 deg. C., of 3.28. The calcium chloride solution, which I suggested (*Trans.*, xiii., 341) for washing bituminous coal, has a maximum specific gravity of 1.40, and is of course not applicable to heavy minerals.*

MR. HOFFMAN: We have experimented at the Tilly Foster mine as regards the magnetic strength requisite for the separation, and we have yet to find (except perhaps an instance of less than 0.2 of 1 per cent.) a material containing magnetite that will not be attracted by a magnet; and we consider that if we had had the distribution on the magnetic separators then as it ought to have been, we would have got practically all the magnetite. Anything that is at all magnetic we can generally recognize at the mine. We do find an occasional piece which looks like hematite, and is not hematite,

* As hornblende ranges from 2.9 to 3.47 in specific gravity (the aluminous varieties being the heaviest), it is evident that neither Thoulet's nor Klein's solution can be depended upon to separate pure hornblende from pure magnetite, though the specific gravity of the latter is 4.9 to 5.2.—R. W. R.

but the proportion of that is infinitely small, perhaps one piece in a month. Our only trouble in separating magnetite from gangue is in having the separators properly fed, the ore properly distributed, and the magnet clean, in good condition and provided with proper air-circulation.

J. C. PLATT, Waterford, N. Y.: Has the non-Bessemer Port Henry ore been brought up to a Bessemer standard as to phosphorus by magnetic concentration on a commercial scale?

N. M. LANGDON, Port Henry, N. Y.: I have not had charge of the concentrating-experiments, but my impression is that the Bessemer limit was not reached. I think that 1.25 phosphorus was reduced to about 0.17.

GEORGE F. KNAPP, Steelton, Pa.: What proportion of the concentrates is carried over into the flues of the blast-furnaces?

MR. LANGDON: I am unable to say precisely, but I should estimate from 4 to 5 tons a month, all told, or about 1 ton a month more than we usually have when concentrates are not charged. On the other hand, the flue-dust is richer in iron when concentrates are used. At the bottom of the "down-comer" it carries about 56 per cent. of iron. This is, of course, returned to the furnace. We are using about 2000 tons of concentrates a month.

DR. DUDLEY: What effect does the use of the concentrates have on the uniformity of the output of the furnace? I ask because of an experience I once had with a furnace concerning which I happened to be called in friendly consultation. It would at times produce the most irregular product; once in a while it would give No. 1 foundry, and then it would jump to a white iron, and there seemed to be nothing in the data connected with the furnace to account for these changes, unless it was the fineness of the ore. As users of pig-iron we find in the shops of the Pennsylvania Railroad Company that one of our greatest difficulties is that a ten-, fifteen- or twenty-car lot of pig-iron will not run anywhere near uniform, or in accordance with the ten-ton sample which we may have bought. In the case of the furnace to which I refer the ore was very soft, almost mud when it came out of the earth, and was allowed to dry before it was put in the furnace. I think it was fossil-ore. The irregularity was mostly in the silicon. Our requirements refer to silicon. We buy a ten-ton lot of iron, analyzing perhaps 2.50 to 2.75 Si, and we put that into a cupola, with a lot of scrap—according to what we want the iron for—and it will carry the scrap very nicely and we will get very good castings. Then we buy a 200-

ton lot, and all at once there is the utmost difficulty in getting along with the castings. An examination of the pig shows a wide difference in silicon. We are making regular determinations from each car-load of anthracite-iron purchased, and we find that the silicon in anthracite-iron runs from as low as 0.45 to 0.50 all the way up to 1.25 to 1.40 per cent. This is in iron specially made for low silicon. It has occurred to me that possibly this use of concentrates might give irregularities in the grade of the iron.

MR. LANGDON: Our ores are the run of the mine. Usually there is a great deal of fine ore, more so, I think, than with other magnetic ores. The use of concentrates has made the furnace-work much more uniform and regular. While we have sudden jumps occasionally from good gray iron to hard iron, they are less frequent than when we used coarser ores, and they grow less the more concentrates we put on. I attribute this result to the fact that as we obtain in the charge a higher percentage of iron by using more of the concentrate, there is less variation in the amount and nature of gangue, and consequently our product is more regular. We make a low-silicon iron, both Bessemer and mill. Our specialty is low silicon.

W. H. MORRIS, Pottstown, Pa.: At the Pottstown Iron Company's furnaces, in making the low-silicon iron required in our basic process, we have averaged for weeks at a time silicon in the iron below 0.4. Sometimes we will run down to a trace, but ordinarily the silicon will fall between 0.25 and 0.6 or 0.7. I think that by paying special attention and care to the handling of the furnaces it is possible to control the silicon. If Dr. Dudley's company would put in their specifications just what silicon they require, I believe that they could very nearly get it. We are also grinding our basic slag so fine that about 90 per cent. of it will go through 100 meshes to the inch, and 60 per cent. of it will go through 150 meshes to the inch. The difficulty we have with the machine we have been using is the steel shot in the slag, which wears out the blades. We have had up the question of magnetic separation, but have not as yet adopted it, and should of course prefer to get along without it if possible. The slag itself is largely composed of phosphate of lime, which makes a very rich fertilizer, and has been used for many years abroad.

E. S. MOFFAT, Scranton, Pa.: Having had some experience in the working of magnetic ore-concentrates, I have read Mr. Langdon's paper with much interest, and to some extent can corroborate

his results by stating that I have never been able to attribute any want of uniformity in the working of the furnace or in the character of the iron produced to the use of concentrates. Our furnaces at Scranton have seemed to work about the same when we were using concentrates as when we were not using them. We have, however, not observed that the concentrates required any less fuel per ton of iron made than other magnetic ores of the same richness, nor have we experienced much increase in the pressure of the blast when using them. If we have not obtained as marked results in these respects as Mr. Langdon has done, it may be due to the fact that although we have consumed a considerable amount of concentrates in the aggregate, we have never used as large a proportion at any one time as he has done.

To the best of my belief the Lackawanna Iron and Coal Company was the first concern to use any considerable quantity of magnetic concentrates in the blast-furnace. We commenced using them about six years ago and have continued their use much of the time since then. Up to October 1, 1891, we had consumed the following amounts of concentrates of different kinds:

	Gross Tons.
Chateaugay,	52,562
Tilly Foster,	21,875
Crown Point,	4,821
Little River,	1,500
Edison,	117
Mount Hope,	40

The Edison concentrates were 50-mesh; the Little River 16-mesh; the Chateaugay $\frac{1}{4}$-inch mesh; and the Tilly Foster $\frac{3}{4}$-inch mesh. Our experience has been most extended with the Chateaugay concentrates, which we have used in varying proportions from $\frac{1}{12}$ up to one-half of the ore-mixture, with results which have been already published. We have used this ore, just as we have used any other ore of like richness, and have noticed no difference in the results. As for the finer concentrates, I would say that our blast-furnace superintendent, Mr. A. H. Lee, has found no difficulty in using Little River 16-mesh, or Edison 50-mesh. These concentrates, being very fine and very dry, have a tendency to fly considerably while being unloaded from the cars and on being dumped into the furnace. This trouble Mr. Lee overcame by moistening the ore before unloading.

We were quite apprehensive that much of these concentrates

might be carried over into the flues, fearing that such extremely fine material while falling from the bell into the furnace and meeting the current of ascending gas would be carried along by it to a considerable extent. We were agreeably disappointed, however, in finding very little in the flues, although what was there was rich in iron, containing over 50 per cent.

MR. KENT: I had the pleasure of visiting the Ogden mine a short time ago, and was impressed with the difference between mining for ore to be concentrated and mining for ore fit to ship without concentration. The old Ogden mine-workings were on narrow chutes or pockets of ore, from 3 to 12 feet wide, and from 200 to 300 feet long, and going down to any depth from which it was economically practicable to pump water. They were interrupted by horses, and by every other trouble known to miners. Now the operators are going over the same ground, and quarrying the whole mass of rock, 300 feet wide and of unknown depth, and they have an enormous quantity of it in sight. In the direct cost of extraction per ton, the advantage of quarrying over the old-fashioned way of mining is immense. The commercial problem of concentration seems to be not so much one of efficient magnetic separators as of quarrying, and transportation to the crushers, and (of more importance, perhaps, than anything else) screening, handling inside of the mill, and getting machinery that will not wear out too fast. The cost of labor is comparatively small; that of repairs to machinery seems to be the most important item at present. I am glad to hear that at the Croton mines they seem to have solved that part of the problem.

MR. RAYMOND: There are two standpoints for the consideration of this subject—the economical preparation of the concentrates, and their economical use. Mr. Hoffman's paper quotes a passage in a letter of mine, published by Mr. Birkinbine in his paper (*Transactions*, xix., 656), on "Progress in Magnetic Concentration of Iron-Ore." This is what I wrote, in connection with other things:

"So far as I can see, at this stage of our practice, waste dumps only, or the rejected portion of an output, other portions of which have been shipped at a profit covering the whole cost of mining, can be used successfully as the raw material of concentration."

In another passage, I said, substantially, that I knew of no American mine where it would pay *to mine and concentrate* a lean magnetite, when there was nothing shipped except the result of the concentration of the ore.

I am willing to stand by those statements still. I do not yet know of any American mine where it will pay, but I admit frankly that the statement presented by Mr. Hoffman in his paper, and amplified by his explanations to-day, indicates such a case; and when the business he describes shall have been continued for a sufficient length of time to enable him to verify his figures from the books of a year's operations on *mined ore* alone, I shall gladly recognize his mine as the exception that proves the rule. In other words, in reply to my sweeping, off-hand opinion that, "at this stage of our practice" (1890), the economical result of magnetic concentration did not leave a profit covering all mining costs, Mr. Hoffman produces a case of actual (but later) practice, in which the business does pay. And, in this case, I notice that the expense of mining, handling, crushing, and delivering the ore, is only 51 cents a ton. This does not include the item of royalty, or any estimate for the condition which ordinarily recurs at such ore-quarries—namely, the necessity of special extra stripping. When that situation arrives, the contractor who has to take out the next 100,000 tons may not be willing to accept the interesting clause in Mr. Hoffman's specification binding him to do the stripping for nothing. That is the time when capital will have to go into the mine, or else the mining cost per ton will have to be advanced.

Again, I notice that a considerable proportion of the material crushed and concentrated, according to Mr. Hoffman's statement, is waste material, already mined; and I am not able, by simple examination of the statement, to deduce from it exactly what would be the commercial costs and profits if all the raw material had to be mined.

Finally, the extraordinary economy he exhibits in the mining and handling of material seems to be due, partly to exceptionally favorable natural conditions, partly to exceptional intelligence and skill in the planning and directing of operations, and partly to large expenditures in machinery and experiments. Of course, it would not be fair to charge against the running cost of mining and treatment per ton, the sum required to recoup, with interest, all experimental outlays. Yet we all know that in every enterprise of this kind, such expenditures are inevitable, though for the pioneers they are heavier than for their imitators. But whatever allowance must be made for experiments, it is certain that the marvellous cheapness of operation, illustrated in Mr. Hoffman's paper, is only to be secured by expensive plants, and by work on a large scale. The mine must

permit such a scale of work, the management must be competent to execute it, and the cost of the plant must be recouped with interest, before the mine is exhausted, or rather, before the *situation* changes, whether that change be due to the actual exhaustion of the mine or the necessity for extra dead-work, or new construction. This charge, like that for the gradual deterioration of plant, is not easy, perhaps, to fix. But the mine-owner has to pay it sooner or later, and not until it has been provided for can real profits be calculated.

I do not wish to be understood as doubting Mr. Hoffman's success, or wishing in any way to detract from it. It does, I confess, sound "too good to be true;" yet I have every inducement to believe it true, and I do so believe. But the pressing question for many of us is, must we simply regard him with sincere and hopeless envy, or can we safely go ahead and imitate his example? So far as it can be imitated, it certainly promises to put new life into iron-mining in this part of the country, giving value to many deposits heretofore unprofitable, and permitting the continuance of work in many localities practically worked out according to the economy of the old method.

The second stand-point is that of the consumer—more particularly, just now, the blast-furnace manager. It seems to me, we have hitherto talked too vaguely about "fine" ore. There are very important differences in the grade of fineness. For instance, there are concentrates which we are glad to get for blast-furnace use, and which give us no trouble. So far as my observation goes, these are usually from machines that leave at least 7 or 8 per cent. (often more) of iron in the tailings, and in most cases, I think, from machines operated with water, which carries off and wastes the finest particles of ore. On the other hand, when the ore-producer, in order to reduce this loss, crushes very fine and concentrates dry, the result may be theoretically a gain to him, but it is not liked by the blast-furnace manager. I think that product will tend to other markets, like the open-hearth furnace, or direct processes, where the conditions of the blast-furnace are not involved.

I will not go into details as to our experience at Durham Iron-Works, or that of Mr. Knapp, at Harrisburg, or what we hear reported about the practice at Bethlehem. The general tenor of it is, that even when moistened, the very fine concentrates give us trouble, whilst the coarser kinds do not. A private letter of Mr. Edgar S. Cook states his experience with them at the Warwick furnace, Pottstown, Pa. Mr. Cook says, in substance, that in 1889 he used

from 600 to 800 tons of the very fine concentrates to form 6½ per cent. of his ore-mixture; that after the first week, the furnace developed irregularities, which diminished or disappeared when the fine ore was taken off, and returned when it was put on again for a week or more; that the amount of flue-dirt was increased. On the other hand, he has since used coarser concentrates (produced by jigging, or by magnetic machines using water) up to 12½ per cent. of his ore-mixture, with very good or fairly good results; but he would rather have crude ore than crushed ore, other things being equal. I think that is a pretty good summary of the general experience so far. "Other things being equal," I interpret to mean an equal price per unit of iron at the furnace, within the limit of say 55 per cent. in richness. That is to say, I think the average manager would rather pay for a 55 per cent. "run of mine" magnetite $3.85 or $4.40 per ton, at the furnace, than $4.55 or $5.20 (that is, 7 or 8 cents per unit in both cases) for a 65 per cent. concentrate. Of course, this is a rough statement only. The nature of the gangue affects the comparison considerably.

Another element of controlling importance in this problem, is that of railroad freights. A mine may be ever so much favored in natural conditions, and yet too far from blast-furnaces to command their custom; and it is the fuel-supply rather than the ore-supply that has hitherto determined the location of furnaces. In the competition with Southern producers, the iron-masters of the North have not been nearly as well supported by the railroads as their competitors. When our railroads shall have realized their duty and interest in this respect, the problem of the commercial concentration of iron-ores will have been completely stated, and we shall have the means of calculating closely what can and what cannot be profitably done. For, as I have observed already, what is to be done in that line profitably, must be done on the largest practicable scale; and hence, each center of production must command a greater area of market than has hitherto been the case. Not merely one or two neighboring furnaces, but a great many that are not so near, must be reached.

AXEL SAHLIN, New York City (Communication to the Secretary): In reply to the question asked at the meeting, whether the Cyclone pulverizer can be advantageously used for reducing iron-ore for concentration, I repeat more clearly what I then said, namely, that, in my opinion, the Cyclone pulverizer would do the work very satisfactorily; but I would not recommend the use of this delicate tool in this case, nor in any other where a machine of lower first cost,

and requiring less power, would crush with sufficient uniformity, and yield a larger output. The Cyclone pulverizer has a field of its own, on products where great fineness and absolute uniformity is required, and for this kind of work it has no competitor.

The importance of reducing to a minimum the wear from abrasion to which a concentrating-plant is exposed, has been pointed out by Mr. Birkinbine and others. This difficulty, greatest in dry concentrating, is materially lessened when the ore is handled wet, the water forming a valuable lubricant. Objections have been raised to wet concentration; the principal one being the expense incurred in drying the concentrates. In my opinion, this objection is of far smaller weight than would at first appear, because, during the mild season, there is no necessity for artificial drying of concentrates not finer than 20-mesh; and during the winter, the concentrates could, if desired, be stored, just as well as the Lake Superior ores are, for shipment in the spring. Even if it be deemed necessary to dry the concentrates, this would certainly be less expensive than to dry artificially the entire bulk of low-grade ore, as I understand, is found necessary in some cases, in order to prepare it for dry concentration.

I beg leave to place before the Institute a few facts referring to the concentrating plant at present successfully operated at the Weldon mines, near Lake Hopatcong, New Jersey.

The separator used at this place is a wet separator, of the Lovett-Finney type, for the description of which I refer those interested to articles appearing in the *Iron Age*, of September 24, 1891, and in the *American Manufacturer and Iron World*, of September 4, 1891, contributed by Mr. Erastus Wiman.

The Weldon ore is magnetic, non-Bessemer, much impregnated with apatite, biotite, and hornblende. The broken ore is assorted at the mine; the purer lumps, stated to run as high as 60 per cent. of iron, are shipped direct; the impure masses, said to average about 30 per cent. of iron, are broken in a jaw-crusher, passed through a set of Krom rollers, then through a 5-mesh screen, and thence direct to the separator.

It seems as if a 5-mesh screen would be entirely too coarse for successful separation of the magnetite from the apatite; and only the very favorable conditions and the open structure of the ore make it possible, with this arrangement, to produce a concentrate which is stated to run from 65 to 70 per cent. of iron, with tailings not above 8 per cent.

My purpose is not to discuss the economical aspect of the work

performed, which, however, is said to be very satisfactory, but of which it is, perhaps, as yet somewhat too early to make a record. I would merely call attention to the fact, that here is a concentrating-plant which has been in continuous operation since the beginning of March, 1891, or for a period of a little over 7 months, during which time it has regularly turned out from 20 to 25 tons of satisfactory concentrates per shift of 12 hours; and most of this time it has been running double-turn.

Up to date, no repairs of any description whatever have been required, except the renewal every eight or nine weeks of the plain canvas belt which carries off the concentrates. This belt costs only five dollars to replace. The breaking and subsequent patching of the cross-head of the crusher can hardly be charged as wear on the concentrating-plant, as it was caused by a pure accident. Besides what has been stated above, there have been absolutely no renewals of any kind, and the chutes, slopes, tanks, boxes, etc., which are made of light sheet-iron and wood, do not show the slightest wear.

In comparison with the large plants at Croton and elsewhere, which have been described during this discussion, the output of this small installation (regulated, however, not so much by the capacity of the separator, which could readily be doubled, as by the output of the mine) is insignificant; but it may be large enough to indicate *one* way of overcoming the embarrassing difficulty of excessive wear on the parts of concentrating-plants.

KENNETH ROBERTSON, Secaucus, N. J. (Communication to the Secretary): I was not at the Glen Summit meeting, and therefore am unfortunately obliged to make my contribution to the discussion with the disadvantage of not having heard what the different gentlemen said, and guided by the meager reports in the trade journals, and the accounts given me by personal friends. As concentrates were used in a blast-furnace five or six years ago by Mr. Moffat, and the fact was published, I had supposed that any discussion of the matter would be ancient history, and in the nature of "threshing over old straw," and am surprised at the quantity of grain which has been gathered therefrom at this late day. Shortly after Mr. Moffat began using concentrates, and before he published the fact, I visited Scranton, and, on going through the stock-houses, was shown the concentrates, and told that they constituted one-half of the ore-mixture, and that they worked well. I was astonished, and could not understand how it could be, for I was, at that time, strug-

gling with a fine ore, and my efforts to use one-half of it had constantly resulted in bad work and possible disaster.

Leaving the stock-houses and going to the tops of the furnaces, I found them equipped with double bells. I immediately thought that I saw, in the better distribution of the stock given by that arrangement, the cause for the successful working of so much fine material. I told Mr. Moffat so, and continued giving that reason for five years, with that "calm serenity of mind" so kindly attributed to blast-furnace men by rolling-mill superintendents (*vide Trans.* xii., 265); and Mr. Moffat did not contradict me. As he had put in the double bells, and had worked the concentrates, the primal success was his, and, from whatever cause it came, he was satisfied with the situation.

About a year ago, it became necessary for us to attempt to use one-half Chateaugay concentrates at the Secaucus furnace. Up to that time we had used not more than one-third in the ore-mixture. With infinite respect for the double-bell theory, and the knowledge that ours was a single bell, I ordered the one-half of Chateaugay material put on the furnace, and with great trepidation waited forty-eight hours for the results. To my great relief, and to the destruction of the double-bell hypothesis, the furnace worked beautifully. The pressure was not increased, nor was the fuel decreased (considering the increased yield of the mixture), but the grade of the iron was improved. After the mixture had been working for two weeks, and making iron which was graded as Nos. 1 and 2, we suddenly, and without any apparent cause, made two casts of white and mottled iron. The burden was immediately lowered, and the temperature of the blast increased, but before the reduced burden came down the furnace resumed its normal working and the former burden was restored. In about ten days thereafter the same thing occurred, but without any change of burden the furnace cured itself, and went on in the even tenor of its way. After this, we frequently used as much as $\frac{5}{12}$ of concentrates without any irregularity in the work of the furnace, and I am disposed to attribute the two slight irregularities before mentioned, when we were working one-half concentrates, to the fact that just previous to our using that quantity we had been cleaning and repairing boilers for a month, during which time it was impossible to give the furnace the quantity of blast which the hearth demanded, and that, in consequence, slight accumulations had formed on the bosh, which came away, without any disturbance noticeable on top, and thus lowered the grade of the iron for a few casts.

During this time the quantity of flue-dust was increased, and, as it contained 53 per cent. of iron, it was saved and filled back into the furnace with the concentrates. As this was not weighed, I cannot tell what quantity we caught in this manner.

As I have said before, Secaucus furnace would *not* work 50 per cent. of fine ore without great irregularity and threatened disaster. It *did* work 50 per cent. concentrates with perfect ease and comfort. The double-bell theory has been demolished to my entire satisfaction. The only explanation I can give is, that the fine ore contained 42 per cent. of iron, while the concentrates contained 62 to 65 per cent. of iron, from which one would deduce that the concentrates worked well because they were rich in iron, while the fine ore gave trouble because it was lean.

If some one will evolve a theory to account for this, it will add to the wheat already extracted from this old straw, and conduce to the happiness and longevity of those furnace-men whose superior officers insist on the use of ore both lean and fine. In this connection, I would like to ask Mr. Moffat, who has worked a greater variety of concentrates than any of us, to give us the percentages of iron which the different ores contained. As he has been successful in working them all, however, this information may not throw much light on the subject; for it appears to be "all grist that comes to his mill."

CLEMENS JONES, Hokendauqua, Pa. (Communication to the Secretary): Mr. Birkinbine's critique (*Transactions*, xix., 656) on "Progress in Magnetic Concentration of Iron-Ore" (a review of the papers presented at the New York meeting, September, 1890), directs attention to one or two minor points in the paper presented by me (*Transactions*, xix., 289), which are, perhaps, capable of clearer explanation.

In regard to the yield of iron-ores, there is a recognized difference between opinions formed by personal investigation and experience, and those based upon mere results of a general nature, which in all cases under my observation are invariably higher than the actual average yield. Obviously, this latter can be determined by systematic examination only. In other words, it is altogether a question of sampling. I have no desire to begin a profitless discussion of such a matter, but only wish to qualify my own statements.

In brown-hematite ores, the size of the pieces of ore as they naturally occur, aided by the character of the *detritus* itself, to which there is no mechanical adherence, largely governs the richness in

metallic iron. It follows that an average of 37 per cent. of iron is made up of a proportionate mixture of large-sized pieces and fine particles. Taking for illustration the figures given in my paper (*Transactions*, xix., 292), we would have:

Three-fourths fine or wash-ore @	34 per cent. iron.
One-fourth large pieces @	46 per cent. iron.
Or an average of	37 per cent. iron.

Moreover, I have determined by experiments in sizing brown-hematite ores of the class described, that this is their average quality, and also that this is very near the average proportion of fine to lump-ore. As a matter of fact, the actual average yield is obtained in a different way, too long to be submitted here. But the distinction of lump-ore and fine, or wash-ore, is a perfectly natural one. My statement is, therefore, that of the 100,000 tons of ore examined, "three-quarters of this was wash-ore," and that the bulk of this (*i.e.*, the fine or wash-ore), representing *this class* of mines in the United States, "will carry about 34 per cent. of metallic iron in the wash ore."

As no "standard" of the exact yield of jigged ores, either general or sectional, was mentioned in my paper, this point is self-supporting; the figures given are based on evidence by no means confined to the broad State of Pennsylvania.

With regard to crushing such ore, my experience has been that it is unnecessary; and accordingly I made no mention of crushing, other than to refer to the decrepitation which takes place during the roasting-process. This is a striking feature of the process, and it stands to reason that, *a priori*, ores in lumps of such size as to require crushing will admit of far cheaper and quicker separation. Other than this, so far as crushing is concerned, *crystalline or granular ores only* are adapted to separation by crushing and magnetism with commercial success. Brown-hematite ores having these properties have not yet been discovered in paying quantities in the United States.

As to the cost of roasting ore, I cannot avoid thinking that Mr. Birkinbine has misunderstood the statement of Mr. C. Linkenbach. Controversies about cost should be confined to the conditions of the case in point. There is not one word in my paper touching the cost of roasting zinc-blende ore. The fact is well known that such ore is extremely difficult to roast, and that intense heat is required, and a specially constructed furnace must be employed if the process is

to be successfully performed. But my paper states that "it is undesirable to use much heat, since at a temperature of cherry-red, magnetization is fully imparted. The ore is drawn as fast as it reaches this temperature," etc. An intense white-heat is usually aimed at in roasting zinc-blende ores. The fuel-consumption given by Mr. Linkenbach is therefore *not* an example of the economical roasting of iron-ore. The abstract in the *Journal of the Iron and Steel Institute* * of Mr. Linkenbach's paper, quoted by Mr. Birkinbine, immediately precedes an abstract from Mr. I. P. Pardee's paper to this Institute (*Transactions*, xv., 678), giving the cost of roasting ore at the Musconetcong works. This was, including cost of labor, fuel, etc., 14 cents per ton of roasted ore.

I trust to be able before long to give the cost of magnetizing and separating brown-hematite ore, as calculated for practice on a large scale.

PRESIDENT JOHN BIRKINBINE: As my name has been mentioned in the papers presented and in discussion, a few words may be in place here. It has been and is still my privilege to act as engineer for Messrs. Witherbee, Sherman & Co., in all their experiments in magnetic separations at Port Henry, N. Y., and I can reply to the query of Mr. Platt, whether the "Old Bed ore," high in phosphorus, had been made into ore of Bessemer grade. When Mr. Langdon stated that up to the present this result had not been accomplished, he spoke truly; but it is fair to mention that when the experiments were carried on we had no great expectations of success, for the trials were made with a mill equipped for coarse grinding, so that it could not reduce satisfactorily to 16-mesh, which was the finest comminution used at any time. The crushing and sizing was done with old, supplemented by new machinery, and therefore the granulation of the ore was unsatisfactory. The firm is now putting up, alongside of the old machinery, improvements such that one-half of the concentrating plant will be equipped with crushers and rolls and the other half with Sturtevant mills. The expectation is to work under similar conditions in either half, so that any ore selected can be put through either side of the mill. In this way we hope to determine conclusively the best treatment for any ore mined by the firm; and with the new machinery in place, we expect to grind to finer mesh than heretofore, and solve the problem whether or not Bessemer concentrate can be made com-

* The original paper is in *Die Aufbereitung der Erze*, Berlin, 1887.

mercially out of ores carrying from one-half to 3 per cent. of phosphorus; for we anticipate that this will be determined in part by the degree to which the material is crushed, in part by the manner of separating the apatite particles from the magnetite, and in part by the efficiency of the separator used.

In the letter quoted I purposely refrained from using the author's name because, as then explained, he was out of the country. I am glad that Dr. Raymond has declared his authorship and reiterated his opinion, for in this I shall follow in his footsteps and attempt to sustain what I said in my paper, namely:*

"The instance cited in the letter, where a 38 per cent. ore could be carried to the furnace and smelted more cheaply than it could be concentrated and then smelted, points either to close admixture of gangue and magnetite, imperfect method, or low cost for fuel and transportation. There are probably few cases where such a lean ore can be smelted to better effect in its natural condition than by first enriching it by concentration, unless the ore possesses in itself other valuable qualities besides the iron. Ore yielding less than 38 per cent. of iron is now concentrated at costs which leave ample margins for mining the crude material, if it occurs under what may be termed average conditions."

I am familiar with the deposit referred to by Mr. Hoffman, viz., the Croton Magnetic iron-mine, having lately made a critical examination of it and reported upon it in connection with our fellow-member, Mr. W. B. Kunhardt.

It is a deposit of magnitude, the ore being magnetite of low grade. A thorough sampling, carried out in connection with the examination, demonstrated that if the entire ore-body were removed, about one-third would be thrown aside as unsuitable for concentration; of the other two-thirds, about 5 per cent. may be rich enough to be shipped as it is mined, but the balance would all have to be concentrated to make it merchantable. The sampling included 380 holes specially drilled and blown, in addition to nearly 10,000 chipped pieces; these, after reduction in volume in the mine, aggregated a total of about 10,000 pounds, which was crushed and quartered, and made up into nearly 400 samples for the laboratory. Such sampling gave an opportunity to judge closely of the occurrence of different percentages of iron in the ore bodies.

The exploited portions of this deposit indicate that there remains within 200 feet of the Theall tunnel level four and one-half million tons of ore averaging 37½ per cent. of iron, and if the workings are carried below 200 feet, or extended to portions of the property where

* *Trans.*, xix., 669.

magnetic surveys indicated a continuation of the deposit, the available amount of ore may be further increased. An ore-body having an average width of 90 feet and extending in length over 2100 feet, inclosed by walls which are nearly vertical, offers excellent opportunity for obtaining ore cheaply. There are no doubt other deposits which, owing to their leanness, have never been thoroughly exploited, but which would compare favorably with that of the Croton Magnetic iron-mine, and some of these unwrought magnetic-iron deposits of lean character may possibly be located as favorably as the Croton mines, which can reach a number of existing blast-furnaces at freight-rates of from $1.50 to $2.00 per ton.

Taking the outlay for labor, supplies, etc., from the pay roll of the concentrating-mill now in operation, it was found that a ton of crude ore delivered at the mill could be crushed, sized and separated for fifty-three cents, and a table was then constructed, based upon the formula given in the paper which I presented at the New York meeting of the Institute.* This table (page 598) shows the theoretical quantity of ore of any percentage of iron between 20 and 55 which is necessary to produce one ton of 65 per cent. concentrate, with a loss of 10 per cent. in the tailings; allowance was also made for 5 per cent. lost in dust, moisture and sulpur.

The table, together with the cost of delivering ore at the separator-building and the cost of treating one ton of such ore, was used to determine the grade of material which could be mined at the Croton deposit, concentrated and loaded on cars without loss. (The estimate included no allowance for interest, royalty, selling-costs, etc.) This limit was found in the case under consideration, to be below 30 per cent. The minimum percentage of iron-ore which could be concentrated without a debit charge against the enterprise will of course vary as to location, peculiar features of the ore and method of concentration pursued. There is no general rule to be laid down to cover all localities, but the plant in practical operation, which although new can be improved upon, offers a basis for estimating the expense of treating a ton of material elsewhere which need not be greater than that named, except in so far as fuel per ton

* *Trans.*, xix., 656. The formula referred to is as follows: The percentage of iron in concentrates (a) minus the percentage of iron in the tailings (b), divided by the percentage of iron in the crude ore (c) minus the percentage of iron in the tailings, is the number of tons of crude ore (n) required to produce one ton of concentrates; or $\frac{a-b}{c-b} = n$.—R. W. R.

Table Showing the Theoretical Tonnage-Ratio of Crude Ore to Concentrate.

Iron in Crude Ore.	Crude Ore required to produce One Ton of 65 per cent. Concentrates, and allowing 10 per cent. loss in the tailings.	
	Without allowance for dust, moisture and sulphur.	With allowance of 5 per cent. for dust, moisture and sulphur.
Per Cent.	Tons.	Tons.
20	5.500	5.775
21	5.000	5.250
22	4.583	4.812
23	4.231	4.443
24	3.929	4 425
25	3 667	3.850
26	3.438	3.610
27	3.235	3.397
28	3.056	3.209
29	2.895	3.040
30	2.750	2 888
31	2.619	2.750
32	2.500	2.625
33	2.391	2.511
34	2.292	2.407
35	2.200	2.310
36	2.115	2.221
37	2.037	2.139
38	1.964	2.062
39	1.897	1.992
40	1.833	1.925
41	1.774	1.863
42	1.719	1.805
43	1.667	1.750
44	1.618	1.669
45	1.571	1.650
46	1.528	1.604
47	1.486	1.560
48	1.447	1.519
49	1.410	1.481
50	1.375	1.444
51	1.341	1.408
52	1.310	1.376
53	1.279	1.342
54	1.250	1.313
55	1.222	1.283

or labor per day would affect it. With a knowledge of what it would cost to remove one ton of material fit for the separator from the workings, it is not a difficult matter to estimate closely the minimum contents of an ore which it would pay to concentrate by magnetic separation at any locality.

The limit will be affected by the degree of fineness to which the ore must be crushed, but in the light of present knowledge and practice and with improved mining and concentrating appliances, it

does not seem over sanguine to expect that with average conditions of cheap mining, low cost of fuel and labor and accessibility to a market for concentrates, magnetic iron-ore carrying 35 per cent. of iron can be successfully mined for concentration and treated, unless the royalty is excessive, or the chemical composition such as to interfere seriously with the sale of the product.

The loss in tailings is largely due to the mechanical imperfection of the separators which have not yet been overcome. We are much further advanced in this than a year ago, and I have faith that we will approach nearer perfection. Whether we can reach a practice which will enable us to pick up one mineral of pure magnetite, leaving another of slightly less magnetic character, is a question which can be decided only by time.

A series of experiments, made in connection with the operation of Messrs. Witherbee, Sherman & Co.'s concentrating-plant at Port Henry, N. Y., suggested to Mr. T. Reed Woodbridge the theoretical investigation of the relative efficiencies of various machines; and by applying to different magnetic separators the formula already given, and verifying the results obtained from actual data, it has been found that the efficiency of magnetic separators in actual use reaches from 65 to 85 per cent. of theoretical perfection. While this is not a complete and accurate measurement, it has sufficient weight to indicate that there is considerable ground yet to be covered in perfecting magnetic separators. Most of the earlier machines were constructed largely as models and without proper ideas of the wear to which they would be subjected in actual use. Any one familiar with practical separation knows how readily and quickly the gangue cuts anything in the shape of metal. I have seen chilled spouts, carrying the ore from the bins to the machine or *vice versa*, cut into regular grooves to the depths of three-quarters of an inch in two or three weeks.

THE USE OF MAGNETIC CONCENTRATES IN THE PORT HENRY BLAST-FURNACES.

BY N. M. LANGDON, PORT HENRY, N. Y.

(Glen Summit Meeting, October, 1891.)

IT is now about two years since we began using concentrates from Port Henry magnetic-ores in the blast-furnaces of Witherbee, Sherman & Co. During this time we have had the usual ups and downs incident to blast-furnace practice, and have used the concentrates in

various proportions with our New Bed (Bessemer), Old Bed (high-phosphorus), and Mill ores, making both Bessemer and mill-iron.

In the early part of 1889, owing to frequent changes in the experimental concentrating-plant, the supply was not regular, causing intermissions in which no concentrates were charged. The tabular statement given below shows the days run, the proportions and kinds of concentrated and other ores used, etc. It may be as well to say that all our ores are magnetic.

The New Bed pure is mostly granular or shotty; the run of mine is usually a lump ore containing more or less of fine. Old Bed is about half lump and half fine, and of the Mill ore $\frac{3}{4}$ or $\frac{7}{8}$ is as fine as buckshot, the rest being lump. All lumps are hand-broken to about 3 inches.

We began using New Bed concentrate in our No. 2 furnace, running on mill-iron, commencing with $\frac{1}{14}$, which was continued for nine days without showing any bad effects, when it was increased to $\frac{2}{14}$, and this proportion was continued with two short intermissions for fifty-six days. The mixture was then changed to Bessemer, and the proportion of concentrate was $\frac{2}{10}$ for twenty-nine days, when it was increased to $\frac{3}{10}$ for fifteen days, then reduced to $\frac{2}{10}$ for eighty days. During this time we had also used $\frac{3}{10}$ concentrate in our Cedar Point furnace, running on Bessemer iron, for fifty-six days.

Up to this time we had not used a larger proportion of the concentrate than $\frac{3}{10}$, but had used it with various proportions of our other ores in making both Bessemer and mill-iron without any more troublesome experience than usual in the working of the furnaces, and also without any advantage, so far as could be observed, except that due to increased yield from the richer concentrate replacing "run of mine" and mill-ores. There was no decrease in fuel-consumption that could be attributed to the use of concentrate except what was due to the increased yield of the mixture. Pressure and general working of the furnaces were about as usual. Our fuel is $\frac{3}{4}$ Delaware and Hudson lump-coal and $\frac{1}{4}$ Reynoldsville or similar coke.

In March last the new concentrating plant at the mines having been completed, it was decided to concentrate Old Bed ore, running about 63 per cent. iron and 1.25 per cent. phosphorus, and at the same time to try the experiment of increasing the proportion of concentrate in No. 2 furnace, then running on mill-iron, with the view of determining the maximum amount that could be used, and to note the effect of its substitution for other ores.

We commenced with $\frac{3}{10}$ concentrate and ran for nine days without noticing any effect either way, then increased to $\frac{4}{10}$ for twelve days. During the twelve days' run, the concentrate was of 16-mesh size, and carried 67 per cent. of iron. All the previous concentrate had been of 10-mesh and 60 per cent. to 63 per cent. iron. When the $\frac{4}{10}$ came through, a change was noticed at once, the furnace began to get hotter and the burden was increased. After the twelve days' run, the concentrate was made 10-mesh in size, and increased to $\frac{5}{10}$ for eight days and then $\frac{6}{10}$ for fourteen days, when all concentrate was taken off.

With each increase of concentrate the furnace got hotter, necessitating increase of burden. It worked very smoothly and regularly, but with 1 to 1½ pounds higher pressure with the larger amount of concentrate. During the latter part of the run, and from the time the charge of $\frac{4}{10}$ concentrate began to work, the average fuel per ton of iron was 18 per cent. lower than it had been for several weeks preceding or was immediately after. Of this reduction in fuel, about 5½ per cent. was due to the increased yield of iron in the mixture.

It is but fair to say, however, that the iron made was less open (about one grade lower) than before the experiment. Just at this time we received an order for a low-silicon iron for immediate shipment; and as the tendency of this mixture, when making an open iron, was towards high silicon, the burden was kept rather heavier than it otherwise would have been, which, of course, made the grade lower and the reduction in fuel somewhat more marked.

After running a short time on mill-iron without concentrate, the furnace was changed to Bessemer, using New Bed concentrate, 10-mesh, 67 per cent. iron, as follows: $\frac{4}{10}$, ten days; $\frac{6}{10}$, four days; $\frac{8}{10}$, fourteen days; and $\frac{7}{10}$, eleven days; total, thirty-nine days. During this time the furnace worked smoothly and regularly, with about the same increase of pressure and reduction of fuel as when using the Old Bed concentrate.

In conclusion, our experience seems to have demonstrated that, with proper management, there is no difficulty in using in the blast-furnace at least 80 per cent. of high-grade concentrate, and that there is also, especially in the use of a large percentage of concentrate, an economy of fuel beyond that which may be due to an increase of iron in the mixture. This increased economy may be attributed to the comminution of the ore.

Table of Mixtures.

Days run.	New Bed.			Old Bed.		Mill.	Iron made.
	Concentrate.	Run of Mine.	Pure.	Concentrate.	Run of Mine.	Ore.	
9	1/14	3/14	.....	.....	4/14	3/14	Mill.
5	3/14	2/14	.....	.....	4/14	3/14	
51	3/14	2/14	.....	.....	4/14	5/14	
29	2/10	2/10	6/10	.....	.....	.....	Bessemer.
15	3/10	5/10	2/10	.....	.....	.....	
7	2/10	6/10	2/10	.....	.....	.....	
73	2/10	8/10	.....	.....	.....	.....	
9	.....	3/10	.....	3/10	3/10	3/10	Mill.
12	.....	3/10	.....	4/10	3/10	1/10	
8	.....	3/10	.....	5/10	3/10	.....	
14	.....	3/10	.....	6/10	3/10	.....	
10	4/10	3/10	4/10	.....	.....	.....	Bessemer.
4	5/10	3/10	2/10	.....	.....	.....	
14	5/10	3/10	.....	.....	.....	.....	
11	7/10	3/10	.....	.....	.....	.....	

PRACTICAL RESULTS IN THE MAGNETIC CONCENTRATION OF IRON-ORE.

BY W. H. HOFFMAN, M.E., CROTON MAGNETIC IRON-MINES, N. Y.

(Glen Summit Meeting, October, 1891.)

THE writer does not claim a right to discuss this subject as a furnace-man or user of iron-ore in this new form. His efforts have been confined to mining, preparing, and separating the magnetic ore from the gangue. Two years' experience in experimenting and another two years in producing concentrates on a commercial scale, have led me to believe that the new trade of making concentrates must be well learned in all of its details. If there have been any partial failures in this most useful branch of the iron business, the failures have been caused chiefly by inexperience or lack of patience. Although I have, within the past five years, constructed machinery

for water-jigging and wet magnetic separation, I shall only speak of the dry magnetic process here.

Three years ago I was engaged by Mr. J. D. Cheever, the lessee of the Croton Magnetic Iron-mines, at Brewster, N. Y., to adapt an old water-jigging mill to the magnetic process. The ore to be treated is described by Messrs. John Birkinbine and W. B. Kunhardt in a late report on the Croton Magnetic Iron-mines, as consisting of compact, moderately fine-grained magnetite in a gangue composed mainly of quartz and hornblende, besides feldspar, apatite, and mica, and more or less pyrite and pyrrhotite, and they say "it is highly satisfactory that the grain of the ore permits a fair disintegration of its constituents by crushing it to pass a 12-mesh screen, inasmuch as a finer mineralization would increase the cost of concentration, as hereafter described."

Similar descriptions have been given by Mr. Emerson McMillan, Mr. F. W. Gordon, and Prof. N. S. Shaler.

By these gentlemen the average amount of metallic iron in the ore-beds is stated to range from 37 to 42 per cent., the average sulphur from 1.7 to 2.2, and the phosphorus from 0.070 to 0.426. Practical work has shown the average amount of phosphorus to be 0.232. Since May 1, 1891, we have been roasting and concentrating this material to 68 per cent. in metallic iron, 0.44 in sulphur, and 0.036 in phosphorus. Previous to this the concentrates ran about 66 per cent. in iron.

Up to the 1st of September, 1891, opening by drifting and prospecting with the diamond drill has exposed at least 8,000,000 tons of ore, while it can be seen by a fair examination of the property that there is not less than three times this amount of ore in these mines, and this estimate does not include the mining-rights owned personally by Mr. J. D. Cheever.

Prof. Shaler says:

"The considerations previously given concerning the horizontal extension of this deposit beyond the limits of the present workings lead me to feel that it is reasonably safe to call the length of this bed 4000 feet, the depth within workable limits 1500 feet, and the width 75 feet. These estimates can be more fairly made than in the case of an ordinary vein, and, though not positively certain, have all the probability they would have if the deposit were a bed of coal. On this basis we have a total of about 50,000,000 tons of ore in the deposit. Of this mass something like three-fourths may be regarded as minable, the remainder being the leaner ores that cannot be profitably extracted at great depths, or ore which must be left in place to sustain the walls and roofs of the mines."

Mr. F. W. Gordon says:

"Very extensive workings and explorations show the deposit of ore to be one of the greatest in magnitude that has ever been developed in America. The explorations and developments have been intelligently made, and warrant an estimate of the quantity of ore in easy reach to be not less than 30,000,000 tons. As the name of the mine implies, the ore is magnetic or Fe_3O_4 in composition and dense in structure. Entries 15 to 20 feet wide driven into the ore require no timbering to support the roof. The average of eleven analyses taken from the workings between June 6th and August 13, 1890, gives the following composition:

	Per cent.
"Metallic iron,	40.64
Phosphorus,	.302
Sulphur,	1.540"

Before the old water-jigging mill was erected, some eight years ago, the product of these mines was sorted to remove the more sulphury ore, and was shipped directly to furnaces making foundry-irons. In order to meet the requirements of purchasers, the mining company was obliged to reject, in cobbing, 2½ tons to get 1 ton of shipping-ore containing 51 per cent. of metallic iron and not more than 1 per cent. in sulphur. About 50,000 tons of this class of ore was shipped from these mines. The old dumps from this sorting are now being crushed and separated by the new process.

Mr. John Birkinbine, in his paper on "Progress in Magnetic Concentration of Iron-Ore" (*Trans.*, xix., 656), quotes from the private letter of a member of the Institute an expression of emphatic doubt whether, at any American mine, it would pay to mine and concentrate a lean magnetite. The writer says: "So far as I can see, at this stage of our practice, waste-dumps only, or the rejected portion of an output, other portions of which have been shipped at a profit covering the whole cost of mining, can be used successfully as the raw material of concentration."

Without repeating here the argument of this correspondent, or criticizing his knowledge of the situation, and without asserting that magnetic concentration was a commercial success three years ago, I can assert and prove beyond question that the Croton Mines have produced and sold at a fair profit from 50 to 220 tons of concentrates per day during the past year and a half; and the mill has been actually running but twenty months.

Commercial success in concentrating any ore includes economical mining, preparation and separation; but in the Croton ore the presence of sulphur calls for very economical roasting as well as economy in all the other processes. The cheap roasting of the ore of the Croton mine, which contains about 2 per cent. of sulphur, was

really one of the first problems encountered in reclaiming that property, and was really as important a factor as the concentration.

A series of experiments was made to determine the best size for economical roasting, and at the end of three months a size that would pass through a 2¼-inch ring was adopted, as giving the most rapid work for the quantity of fuel consumed. Crude Lima oil is used for roasting. Through experiments conducted by our general foreman, Mr. T. Blass, we found the average consumption of fuel-oil to be 3.75 gallons; but by enlarging the combustion-chambers we have reduced this amount to a little over 3.6 gallons per ton of raw ore. The cost of the oil is 2½ cents per gallon, making a fuel-cost of 8½ cents per ton of raw ore. The labor of filling and discharging amounts to only 3 cents per ton, as this work is largely automatic. The average temperature is 1250° Fah. Davis-Colby roasters, remodeled to burn fuel-oil, are used for a portion of this work, the remaining portion being done in a roaster of new type designed by the writer. The Davis-Colby roasters have been in operation nearly three years, and have done excellent service.

The ore is conveyed automatically from the roasters to the Sturtevant mills, where it is ground to 12-mesh size, all coarser material from the screens being returned to these mills by elevators. Barring the numerous experiments with various types of magnetic separators, the experiments in crushing have been the most elaborate. Nearly all the best-known methods of grinding ores have been tried at the Croton mines during the past three years. Some of these machines have been tested for a year or more, but about fifteen months ago we became thoroughly satisfied that the Sturtevant mill was far superior to any other machine for *grinding* iron-ores. I consider it necessary to mention this machine thus somewhat prominently, as its economical and uniform granulation plays an important part in answering the question in Mr. Birkinbine's paper: "Does magnetic concentration pay?" If the ore is not properly granulated and screened, no known method of separation or concentration can make it a commercial success.

The screen-block openings in the Sturtevant mills are ¼ inch wide, and the coarsest material passing through them is less than $\frac{7}{32}$ of an inch thick, while the finest material would be rejected by a 60-mesh screen.

The ore enters the Sturtevant mills at a temperature of about 350°, being cooled from about 1200° by a water-bath on its way up the conveyer. Under these conditions the ore is quite friable, and

we have no difficulty in grinding 22 tons per hour with the 20-inch mill, and 16 tons in the same time with the 15-inch mill. One set of Sturtevant-mill bushings will grind from 4000 to 6000 tons of ore, according to the depth of the chill in the bushing, the cost of each set being $16.00. The screen-blocks for this amount of ore cost $9.00. This is less than one-half the cost of renewal on any other machine formerly used at our mill. At 22 tons per hour the 20-inch mill requires 94 horse-power to drive it, but it will be remembered that the product is finished on these mills. The 15-inch mill requires 70 horse-power.

The ground ore is elevated from the discharging-nozzles of the Sturtevant mills to the several screens, covered with slotted steel plates made by the Harrington & King Perforating Co. The slots are $\frac{1}{12}$ by $\frac{1}{2}$ inch in some plates, and $\frac{1}{12}$ by $\frac{5}{8}$ in others.

The slotted plates are easily removed, and when the requirements are exacting as to phosphorus, we substitute plates of $\frac{1}{14}$ mesh on two of our five screens. We have demonstrated by exhaustive experiments that two sizes of screen-plates, three sets coarse and two sets fine, will prepare the ore containing 0.426 in phosphorus (the greatest amount we have in the mine) for a separation having 0.036 with two passes on the magnetic separators. Ordinarily, the phosphorus in the Croton ore runs from $\frac{1}{10}$ to $\frac{5}{10}$. When the phosphorus runs higher than $\frac{6}{10}$ three sizes of screen-plates should be used, delivering to three receiving bins, and each size should be treated separately on the magnetic separators; and I am positive that this treatment will insure a Bessemer product running not over 0.050 in phosphorus, using nearly any of the New York State magnetites that are free from titanium.

In some experiments we have used 18-mesh screens, and with ore prepared for this grade we were enabled to produce continuously, with two passes, concentrates showing 70.60 metallic iron, 0.018 phosphorus and 0.220 sulphur. Of course the silica was extremely low. With the latest Hoffman separator, using 12-mesh screen, and making two passes, we have produced concentrates showing 70.93 metallic iron, 0.017 phosphorus and 0.231 sulphur; and by using 18-mesh screens we can depend on 71 per cent. concentrates with one pass on this machine.

Examinations for the purpose of detecting screen-, shute- and bin-leaks must be made at least twice a week.

The screens deliver their finished product to two bins placed on the floor above the separating department, each having a capacity of

80 tons. Eight shutes deliver the ore to the separators, nine in number. The separators were designed and constructed by W. D. Hoffman and T. Blass, at our mines. On eight of the machines the ore is passed before the magnets twice to bring the loss in tailings to 8 per cent. Our Superintendent, Mr. W. D. Hoffman, lately designed two new separators, one of which is giving with one pass concentrates of 68 to 70 per cent., with a loss of only 6½ per cent. of iron in the tailings. As patents are now pending, these machines cannot be described here. People frequently ask how much it costs to separate the iron-ore from the gangue. We always reply, as little as any other portion of the process. Seven cents per gross ton of concentrates is a liberal allowance, and this includes all repairs to separators. We shall reduce this to less than six cents soon. Repairs and supplies throughout our milling-plant amount to one and eight-tenths cent per ton of raw ore ground. Hence, it will be readily seen that *the* feature of the problem of magnetic separation is the initial mill-grinding or granulating. The very cheapest process is absolutely necessary, as most of the ores to be reclaimed will analyze less than 40 per cent. metallic iron, and that means from two to three tons of ore to produce one ton of concentrates. Using Sturtevant mills, and 22 per cent. ore, we can pay a small profit at our plant. Crushers and rolls require 28 per cent. ore to pay at the present price of concentrates. We contract our mining and initial crushing ready for, and delivered to roasters, to Mr. Charles Vivian, of Brewster, N. Y., at an average price of $1.38 per cubic yard. The ore weighs from 5500 to 6800 pounds per yard. We are selling concentrates at present to six furnaces, which use from 35 to 53 per cent. of them in their regular mixture. The furnacemen tell us that their flux and fuel are reduced, but most of them decline to give the exact amount. We have yet to receive a complaint from any of our customers as to quality or fineness. I have personally witnessed the use of several hundred tons of our concentrates at the experimental Ramel Conley steel-works, near our mines. They were used in various ways; sometimes loosely thrown in the bath, and at other times made into briquettes and charged in the open-hearth furnace along with the piles of scrap or African ore.

Every one of the practical open-hearth melters employed there at different times has informed me that he was surprised at the rapidity with which he could handle the furnace when using concentrates, and I understand this has been the experience of others. There is no difficulty found with the blast at Scranton, where concentrates have been used by Mr. E. S. Moffat for several years.

An accurate, though condensed, statement of the cost of mining, crushing, roasting, preparing, and separating one gross ton of 68 per cent. concentrates, from two and one-fifth tons of 38 per cent. ore, according to the present daily practice at the Croton mines, is given below. The tailings run, in iron, from 7 to 8 per cent. About one-third of the ore is taken from the old dumps. On the basis given below, 580 tons is crushed every 20 hours, with a production of about 265 tons of concentrates in the same time:.

Statement of Cost.

Mining, crushing, and delivering to roasters 2⅛ tons of raw ore, at 2¼ gross tons per yard,	$1.13
Roasting, including top-filling,	.23
Handling at roasters,	.03
Preparation and screening,	.22
Daily renewals, supplies, and repairs of all machinery and roasters,	.05½
Separating, including labor and power,	.07
Delivery to Harlem R.R. switch, including R.R. repairs,	.04
Office and laboratory expenses,	.04½
Insurance, interest, and taxes on plant,	.13
	$1.95

During the last winter and spring, a continuous run of five months was made of twenty hours each day, and the average cost of a gross ton of concentrates for the whole term was $2.10.

Improvements have reduced this amount to $1.95, as shown by the figures given above.

Analyses are made at our laboratory daily of ore at mines, ore after leaving roasters, concentrates and tailings, by Mr. G. K. Volckening, Jr., our chemist.

The following analyses cover average two weeks' shipments in July, August, and September of the present year. All concentrates were from screens with slots $\frac{1}{12}$ x ½ inch, and all samples were from car-loads:

	July 7.	July 15.	August 27.	Sept. 5.
Metallic Iron, dried @212° F.	68.55	68.080	68.250	70.090
Phosphorus, " "	.0365	.0368	.030	.045
Sulphur, " "	.240	0.340	.475	.170
Silica, " "	3.500	3.610	2.951	2.513

The average Bessemer concentrates for a period of eight months contained:

	Per cent.
Metallic iron,	66.78
Phosphorus,	.054
Sulphur,	.468

and the average Foundry concentrates:

Metallic iron,	65.29
Phosphorus,	.069
Sulphur,	.724

THE DETERMINATION OF IRON IN THE TAILS FROM MAGNETIC CONCENTRATION.

BY E. K. LANDIS, POTTSTOWN, PA.

(Glen Summit Meeting, October, 1891.)

THE question of magnetic concentration is assuming considerable importance, and the efficiency of the different concentrating-machines is widely discussed. As the amount of iron left in the tails must vary with commercial requirements, and with the nature of the minerals in the tails, the writer feels justified in calling attention to some points often overlooked.

In almost every such commercial operation a point is reached beyond which it does not pay to go, the material saved by further operations being obtained at a cost too great. The amount of iron left in the tails from concentration must depend, therefore, upon the cost of crushing, the size to which the ore is crushed, the nature of the minerals in the gangue, and the nature of the ore itself. Some ores are so intimately mixed with their gangue that a large loss of iron in the tails is unavoidable. In many cases the gangue contains hornblende, pyroxene, serpentine, etc., which carry considerable iron as silicate, or otherwise combined, and this iron a magnetic machine cannot extract, since the minerals themselves are not magnetic. It is evidently unfair to charge the machine with this loss of iron; and hence a satisfactory test of any such apparatus would require that the iron present in the ore as magnetite should be determined, and the proportion in the tailings of such iron only should be the measure of the loss, as compared with theoretical efficiency.

This would be an easy matter, if the gangue consisted only of minerals insoluble in hydrochloric acid, such as pyrites, hornblende, pyroxene, feldspar, quartz, mica, etc. But minerals soluble in hydrochloric acid are often present, such as serpentine, chondrodite, brucite, enstatite, chrysolite, helvite, ilvaite, lepidomelane, etc., not to mention minerals partially soluble, like hypersthene, acmite, garnet (some varieties), epidote, iolite, lepidolite, glauconite, etc. Moreover, the amount of iron in all these minerals varies with different varieties of the same species, which complicates the problem still more.

At the Croton mines, near Brewster, New York, the magnetite was easily determined by boiling the tails with concentrated hydrochloric acid, the gangue consisting of quartz, hornblende, feldspar, mica, and pyrites, with some pyrrhotites, of which the only soluble mineral was the pyrrhotite; and this being magnetic, should have gone into the concentrate and not into the tails. It seemed proper, therefore, to consider all the iron soluble in hydrochloric acid as coming from magnetite. The tails from the ore of Tilly Foster Mine, near Brewster, New York, on the other hand, are partly composed of minerals, besides magnetite, soluble in hydrochloric acid, and containing, according to Dana's *Mineralogy*, the following percentages of ferrous oxide: serpentine, 10.87; brucite, 5.63; chondrodite, 5; enstatite, 10. They contain also other similar minerals, so that boiling hydrochloric acid extracts considerably more iron than that existing as magnetite.

As evidence that the difference between the total iron and that extracted by boiling hydrochloric acid, even under the above conditions, is considerable, the following analyses are given by permission of Mr. E. S. Moffat, general manager of the Lackawanna Iron and Steel Company.

Record of analyses made during six months of 1891:

Total Iron.	Iron Soluble in HCl.
12.993	10.954
11.299	8.005
11.173	9.550
9.216	7.344
10.670	8.238
9.164	8.096
9.773	8.380
10.339	8.498
11.614	9.773
10.985	9.013

	Total Iron.	Iron Soluble in HCl.
	12.500	9.140
	10.653	7.670
	12.394	10.281
	12.676	10.844
	10.240	8.027
	11.204	9.663
	9.383	7.842
	10.223	8.823
	11.064	8.963
	10.795	8.664
	13.494	11.647
	11.931	9.800
	11.647	9.374
	11.516	9.690
	11.095	8.988
	12.078	9.971
	12.640	10.954
	13.904	11.235
	13.482	10.814
Average,	11.384	9.318

The amount of iron existing as magnetite in the tails has another aspect. At present, commercial considerations prevent too close a cleaning of the tails; but these conditions may be modified or may entirely disappear in the future, so that tails now rejected, and carrying 15 to 18 per cent. of iron, may be worked at a profit, the material being already crushed. When this situation arrives, the importance of knowing how much of the iron exists as magnetite, and how much is present otherwise combined, and therefore beyond recovery by magnetic machines, though it might not be very great in the case of tails like those of the Croton mines, might prove a serious element in considering the treatment of such material as the Tilly Foster mine produces.

NOTE ON SAMPLING IRON-ORE.

BY E. K. LANDIS, POTTSTOWN, PA.

(Glen Summit Meeting, October, 1891.)

In connection with the interesting paper of Mr. Glenn on "Sampling Ores Without Use of Machinery" (page 155 of the present volume), I venture to offer the results of ten years' experience in the sampling of iron-ore by a method adopted when I first became

engaged in the analysis of ores, and continued to the present time. This method seems simpler than the one described in Mr. Glenn's paper, and it has proved, for iron-ore, quite accurate, as checked by samples taken according to more elaborate methods, and analyzed by such chemists as McCreath, Booth, Garrett Blair, and others, with results rarely differing more than 0.1 to 0.2 per cent. I regard it, therefore, as not less satisfactory, while it is much more rapid than other methods.

If a pile or car-load of lump iron-ore is to be sampled, each piece that can be reached over the entire surface is picked up, and a small fragment is broken off and kept, the size of this fragment being governed by the size of the lump from which it is broken, that is, a larger piece being taken from a large lump than from a small one. In case the lump consists of ore with adhering gangue, a piece of each is taken, the size depending, as before, upon the size of the lump and the relative amounts of gangue and ore in it.

When the entire surface has been gone over, the sample is reduced by crushing until it all passes a 10-mesh sieve. It is then thoroughly mixed on a large sheet of hardware-paper, first with a coal-shovel, then by turning over and over from end to end, and from side to side. When thoroughly mixed, a small portion is taken with a large spatula from points all over the heap, and this is reduced in a Riotte mortar until it all passes a 20-mesh sieve. It is then again treated on the paper as before, and the resulting sample is passed through a 40-mesh sieve and bottled. This is subsequently mixed once more on a smaller paper, and a sufficient quantity for analysis is taken out and ground to the required fineness in an agate mortar.

The above procedure is employed, as already observed, for lots consisting entirely of lump-ore. When fine ore mixed with lump is to be sampled, the lump is sampled first as above; then the fine is sampled by taking pinches or handfuls from all over the exposed surface, the quantity taken being such that the same ratio is observed between fine and lump in the sample as exists in the ore itself.

While the element of judgment involved in this method creates an undoubted source of possible error, and makes such error easy through ignorance, carelessness or intentional dishonesty, the practical question is, whether the method, intelligently used on such material as iron-ore, actually gives trustworthy results. An excellent test of this question is afforded by the sampling and analysis of crude ore,

concentrates and tails carried on for four months at the Tilly Foster mine. The crude ore was sampled as above described. Of the concentrates, cores were taken in eighteen different places on each carload by driving through it a 1½-inch tube, slightly swedged in at the lower end, like a "choke-bore" gun. The tails were sampled by collecting four bucketfuls from each spout, allowing them to settle, decanting the water, and drying the residue.

The weights of crude ore and concentrates were accurately known, and it is evident that by applying to the results of the analyses of the samples the formula given by Mr. Birkinbine (*Trans.*, xix., 673), a figure representing the number of tons of crude ore required to produce one ton of concentrates could be obtained, which, if sampling and analysis were correct, would correspond with the direct determination in practice. In the discussion of Mr. Glenn's paper, already cited, Dr. Raymond has applied this test to reports of concentrating-experiments, arguing from the discrepancy between the results of the formula and the reported weights actually used and produced, that there must have been defective sampling for analysis.

Such a comparison was made for the operations of four months at the Tilly Foster mine, with the following results:

	I.	II.	III.	IV.
Factor directly found from weights, . .	2.52	2.360	2.73	2.760
Factor calculated from analyses, . . .	2.43	2.292	2.66	2.626
	0.09	.068	0.07	0.134

This agreement is as close as could be expected; and the almost uniform difference in one direction points rather to loss of material in handling than to defects in sampling or analysis. The latter might be expected to give variations in either direction.

THE PREPARATION AND UTILIZATION OF SMALL SIZES OF ANTHRACITE.

(A Discussion at the Glen Summit Meeting, October, 1891.)

ECKLEY B. COXE, Drifton, Pa.: Anthracite coal differs from other fuels in its greater solidity. It does not burn like coke or wood, or like bituminous and coking coals, which become more or less spongy when heated, so that combustion can take place to some

extent within the mass of each piece. Anthracite burns only at the surface. The air-passages which are necessary for an anthracite fire can only be secured between the lumps, piled in contact with one another. It follows that the coal must be separated into such sizes as will leave proper air-spaces between the lumps, and that lumps of many different sizes must not be mixed in burning. For example, stove-coal (the pieces of which pass through a mesh two inches square) will burn beautifully in a stove; but, if it be mixed with a little chestnut and pea-coal, these smaller coals will fill up the spaces between the larger lumps; and, although a greater aggregate surface of coal is exposed, not enough air passes through to produce perfect combustion. One of the most important points in the preparation and use of anthracite coal is good sizing. While this is universally admitted, I do not think sufficient attention has been paid to it in attempting to burn the small fuels. I have found by repeated experiment, to my great surprise, that a certain coal, the large sizes of which (broken and egg) burned badly under boilers, gave me, when burned in the form of buckwheat or pea, better results than any other of our coals. The explanation was, that when we burned egg or broken coal, the ash, instead of falling away, adhered to the coal, and there was formed ultimately around each lump a layer of ash, not to be detached without a great deal of shaking, which interfered very seriously with the fire; and at last the layer became so thick as to prevent the oxidation of the carbon underneath. It was a very pure coal, but the ash adhered to it. But when we burned the buckwheat of the same coal, the small unit-lump was consumed before the layer of ash became thick enough to preclude combustion. With our No. 1 wet buckwheat from that mine, we evaporated 7.78 pounds of water per pound of coal as against 6.79 pounds with No. 1 Drifton buckwheat, which we consider to be our best steam-coal. In fact, this coal, which had proved so unsatisfactory in larger sizes, evaporated almost as much water, when burned as No. 1 buckwheat, as did the Drifton pea-coal. This shows incidentally that a test of the evaporating capacity of one size may be misleading for another size of the same coal. In other words, the fact that you are familiar with the larger sizes of coal from any particular colliery may mislead you as to the value of the smaller sizes.

Originally only the large lumps of anthracite were shipped to market; then broken-, then egg- and then stove-coal were introduced; and afterwards chestnut, pea, buckwheat and the smaller

sizes were utilized. As a general rule, with the exception of what is known as "pea-and-dust," the smaller sizes came into use first as boiler-coal at the collieries. "Pea-and-dust" is the screenings made in shipping coal at the docks into vessels. It is very pure, consisting of fragments from the larger pieces of pure coal. This was bought, for a number of years, at a very low price at the shipping points, by people who were intelligent enough to use it. The price, for a long time, was less than the freight on the larger coal. The material was used either alone or mixed with bituminous coal. This use of the pea-and-dust called attention to the immense amount of fuel, of apparently as high quality, in the culm-banks at the collieries. The coal from the culm-banks was first burned in large quantities in the vicinity of Scranton; and its use is increasing. But a marked difference was soon realized, between the pea-and-dust of shipping-ports and the material from the culm-banks. First, the banks, in many cases, contained large amounts of slate; secondly, the culm itself contained a much larger percentage of slate than the pea-and-dust; and thirdly, where these banks had been exposed to the action of the weather for any great length of time, the fuel had deteriorated.

It should be noted also that the old culm-banks are much richer in pea and buckwheat than the more modern ones. Some of the older banks contain almost all the pea made at the collieries, while the culm-banks of the present time generally represent a refuse, from which not only the pea but also the No. 1 and No. 2 buckwheat have been taken out. This distinction is very important. Our experiments have shown that if we attempt to burn, upon any grate which we now know, culm from which all these sizes of coal have been extracted, a large portion of the fine coal remaining will run through the grate into the ash-pit, after the fuel has become dry. But if there is a certain proportion of larger sizes of coal mixed through the dust, one of the larger pieces will very soon run into the opening and stop it. It is therefore very important, in arranging to burn culm, to know the exact proportions of the smaller-sized coal mixed through it. We have also found that in burning No. 2 or No. 3 buckwheat, it is important to have the finer dust, as far as possible, taken out of it, since this goes into the ash-pit without burning, and chokes the draft. It is really as easy (or easier) to burn the very smallest coal, free from dust, as the next larger size containing dust. We have burned successfully coal that goes through a round hole three-sixteenths of an inch in diameter and over a

round hole one-sixteenth of an inch in diameter, when the coal was clean.

Knowing that some of the people who bought pea-and-dust in New York mixed it with bituminous coal, we procured some highly bituminous coal, mixed 10 per cent. of it with our No. 3 buckwheat, and burned it carefully, weighing the coal and the water evaporated. We found that we got better results from the No. 3 buckwheat alone than we did when we mixed the 10 per cent. of bituminous coal with it. My explanation is, that the finer particles of bituminous slack closed the air-passages to a certain extent, while, on the other hand, this coal was not present in sufficient quantity to produce a coking-effect. The mixture evaporated 6.54 pounds of water per pound of fuel; while the No. 3 buckwheat alone evaporated 6.60. The men decidedly preferred to fire with the unmixed coal.

In the preparation of these smaller coals, we have found great advantage using what is known as the feldspar-jig, having the bottom covered with pieces of feldspar, through which the impurities pass down. We have been very much astonished to find how much iron pyrites is present in the smaller coal; in some cases over 50 per cent. of the jig-refuse. The cause is, that the pyrites occurs generally in small flakes in the coal and slate, and breaks up into very fine pieces, which find their way into the very small coals. I am satisfied that for this reason careful jigging, as well as careful sizing, is necessary for their economical use. The determination of iron pyrites was first made by taking samples of jig-drawings, screening them into several sizes and picking out the pyrites by hand. This being a very laborious operation and hard upon the eyes, my assistant, Mr. Wagner, suggested the extension of a plan originally proposed by Dr. Drown (*Trans.*, xiii., 341), namely, the separation of the slate, iron pyrites and coal by the use of sulphuric acid, diluted so as to form liquids of different specific gravities, upon one of which the coal would float, on another the slaty coal and on a third the lighter slate; what falls to the bottom of the heavier solution being practically iron pyrites and heavy slate, which can then be easily separated by the eye.

In many cases, the amount of slate in the smaller sizes bears no relation to the amount of slate in the whole vein, or in the larger sizes. Some slates are tough and hard and do not break much into small pieces. When the slate in the vein is of this character, the smaller coals will be comparatively clean. But where the slate is friable and breaks up easily, a much larger percentage will be found

in the small sizes. I have already referred to the importance of good sizing in order to secure good combustion. It is equally important for good jigging. For these two reasons, we make three sizes of buckwheat, which enables us to jig them well and to burn them to advantage after they are jigged. The feldspar-jigs have the advantage of removing, in the jigging-operation, a large proportion of fine coal which the screens do not separate, but which should be taken out. We use for our larger coals, down to pea-size, piston-jigs with automatic discharges for coal and slate (a modification of the Lührig jig). But, for our smaller sizes, these jigs do not seem to work well, and our experience indicates at present nothing better than the feldspar-jigs, which are used very largely for small sizes in Europe. Descriptions of them can be found in almost all treatises upon coal-washers abroad. We cannot, of course, get out all the slate; but we reduce the percentage of ash very materially, and get rid of the heavy slate and iron pyrites, which are the most troublesome impurities.

Let me now say a few words as to the boilers. Originally the fuel burned at the mines had practically little or no market-value; capital was difficult to obtain; and operators built their boilers with the view of getting the greatest amount of steam with the least investment of money in the plant, the amount of coal burned not being considered at all. The boilers were plain cylinder-boilers, and generally the products of combustion from all of them ran into a single stack, which, by the natural increase in the number of boilers as the colliery became older and larger, often became insufficient; so that in many cases more money was spent for the plant than would have been necessary had proper stacks been built for the boilers. For $10 in stacks, $50 was wasted in building extra boilers. There has been a marked improvement in this respect. The tendency now is to put up a stack for each nest of boilers. Originally, when large coal was used, simple grate-bars were used; but, as the use of small coals increased, improved forms of grates had to be adopted, since the ordinary bars could not be placed close enough together to let the air through and yet support the coal. The first improvement was to make the bars in two parts: a lower part, which did not reach the fire, but carried the weight, and an upper bar, which was made in small sections and supported the coal directly. The oscillating bars of different types, of which the McClave bar is an example, began to be used. These are found to be generally necessary for the very small coals, although in some cases such coals are

burnt successfully on plates with small holes in them. At about the time of the introduction of the movable bars, the use of a forced draft produced by a steam-blower came into successful use, and seems to have produced very good results. In some cases, fan-blowers are used. The steam-jet, while perhaps not as economical theoretically as the fan, has other advantages. We have less trouble with the clinkers; this, I think, is due partly to the cooling action of the steam and partly to the fact that the steam is decomposed, and carbonic oxide and hydrogen are formed when the steam reaches the hot fire, thus reducing the temperature near the bars, where the clinker is likely to form. Where clinker forms in large quantities, the oscillating bars are difficult to manage; and I found on one occasion that, by putting a very powerful blower in the stack, we got an excellent fire, but our grate-bars soon became so clogged with clinker that it was almost impossible to clean the fires; and we abandoned the use of it, preferring to blow the air in by means of a steam-jet under the grate-bars.

We have also observed that, not only at our own collieries, but at many others in the coal-region, the escaping gases are entirely too hot, often exceeding a thousand degrees. To remedy this, we began by putting feed-water heaters in the stack. We then put mud-drums under our cylinder-boilers, thereby increasing the evaporation 41 per cent. without increasing the quantity of coal used or the labor in firing; and lately we have put in, between our stacks and the cylinder-boilers with mud-drums, a boiler made entirely of pipe, which has again increased our evaporation about 40 per cent., and reduced our stack-temperature to about four hundred degrees, or a little over. Of course, if we were erecting an entirely new plant, it would probably be easy to devise a better system of boilers than those we use; but companies having the large plants now in operation in the coal-regions, cannot afford to throw them away, and must endeavor to make them more economical by gradually improving them. The great point is to keep continually working towards something better, without spending so much money as to hamper the business.

I would like to insist again upon the importance of having the coal as free from dust and smaller sizes as possible, in order to obtain good results. A great deal has been said and written about the immense amount of coal in the culm-banks. I think the amount is exaggerated. Many of these banks are spoilt by being mixed with other refuse; many of them have deteriorated so as to have com-

paratively little value; many have been on fire; and finally, banks when opened turn out, in many cases, not to be as large as they were expected to be.

I am informed that, while there has been since 1883 a great increase in the shipments of anthracite, the whole of the increase, and perhaps more, is in sizes below chestnut, showing that there is a gradual but steady progress in the use of these small coals which were formerly regarded as waste. I think the gain by reduction of the stack-temperature, to which I have referred, is a matter to which too much attention cannot be given. Another point too little considered by the users of coal, is the great economy which might be secured, not only by manufacturers but by coal-operators themselves at the mines, by the use of high-class economical engines. We are using with great success automatic cut-off engines, compound engines, and (wherever we have the water to do it) condensers; and we have proved, by many experiments, their great economy, not only in running cost, but also in the installation of the boiler-plant.

E. G. SPILSBURY, Trenton, N. J.: We have recently experimented at the works of the Trenton Iron Company to determine the relative economy of oil as a fuel, compared with No. 2 or No. 3 buckwheat. It may be pleasant for you gentlemen of the anthracite regions to hear that, so far, buckwheat is ahead. The evaporation we have been able to get, not in any test of a few hours, but on the average for five months, has been (deducting the time consumed in cleaning fires) about 9.7 pounds of water per pound of buckwheat coal. This is stated in terms of the water as we fed it to the boiler, and of the pressure of steam used. Our water goes into the boiler at an average temperature of about 160 degrees F., and we carry an average steam-pressure of 95 pounds. We use the upright Manning boiler altogether. The fire-boxes are 8 feet in diameter, and the tubes are 18 feet long. Our stack-temperature averages about 460 degrees F.

MR. COXE: Of course, that is the decisive test. If your combustion is complete, and your heat does not go out of the chimney, it must go into the water.

MR. SPILSBURY: The cost of oil to us is about 2.1 cents per gallon, and that of the coal, delivered at the boilers, is about $1.94 per ton. I think it is No. 2 buckwheat. On that basis of cost, the economy is as seven to five in favor of the buckwheat anthracite. We have also tried mixing with bituminous coal, with results confirming the experience of Mr. Coxe. We got no practical advantage,

but the contrary. We began with half-and-half, and went up to 75 per cent. bituminous, with 25 buckwheat. Combustion was good for a short time, while the fires were fresh; but afterwards the buckwheat would pass out unconsumed, being occluded in the ashes of the bituminous coal. We find one objection to the use of the buckwheat anthracite, namely, the dust which blows out of our stack.

R. J. FOSTER, Scranton, Pa.: I will try to furnish, as a contribution to this discussion, a description of the McClave grate and its use in Scranton for the burning of culm, to which Mr. Coxe has alluded.*

E. S. MOFFAT, Scranton, Pa.: The ordinary mixed culm, as used in the works at Scranton for steam-purposes, contains all sizes under pea-coal. The steel-works and rolling-mills of the Lackawanna Iron and Steel Company make all their steam with this fuel; and its use has proved a great economy. In running full time, the consumption of culm at our north and south Scranton works amounts to more than 15,000 tons monthly. The use of this liberates for market nearly as large a quantity of coarser coal, which we would otherwise be obliged to burn. As the culm costs only 20 to 25 cents delivered at the works, the greatness of the saving is apparent. The current production of culm from the mines being large, we have not used much from the accumulated piles, but have run chiefly on fresh culm; and we get rather better results from it than from that which has been long exposed to the atmosphere.

The successful use of mixed culm under boilers was accomplished at the mills of the Lackawanna Iron and Coal Company more than nine years ago. Mr. T. G. Wolf, the superintendent, and Mr. B. Niebell, the chief engineer, had experimented considerably with various forms of stationary grates without much success; but when they adopted the shaking grate, combined with the steam-jet blast, the problem was solved. The main trouble was, that in firing culm on stationary grates the fire was torn to pieces whenever it was necessary to clean it, thus occasioning much loss of time; but by the use of the shaking grate the ashes could be removed from below, without materially disturbing the upper part of the fire.

The Howe grate, first used, was soon followed by the McClave, which was adopted by the Scranton Steel Company. Both are now in successful operation at the Lackawanna Iron and Steel Company's works. As for the evaporative power of ordinary mixed culm,

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* NOTE BY THE SECRETARY.—See Mr. Foster's paper on this subject, in the present volume, page 628.

practice with our boilers shows an evaporation of 6.4 pounds of water to 1 pound of fresh culm. We estimate that our culm costs us not over 2 cents per horse-power per day of 24 hours. The labor of firing and taking out ashes and the repairs are no greater with culm than with coarse coal.

The Secretary read the following communication:

F. M. F. CAZIN, Hoboken, N. J.: So far as the making of steam is concerned, the utilization of anthracite smalls was practically solved for me long ago, by burning $\frac{2}{3}$ anthracite with $\frac{1}{3}$ bituminous smalls on the well known *Treppenrost*, or step-grate, in connection with an inclined boiler (manufactured by Henschel & Son, Cassel, Germany).

I believe I was the first to practice another method for utilizing this fuel. In 1854 and 1855, I owned and worked at Schledehausen, near Boomte, in the principality of Osnabrück, Germany, a coal-mine on a bed belonging to the Wealden formation. The coal had the chemical composition and appearance of anthracite, but, upon exposure, it crumbled totally to smalls. Owning also extensive deposits of bog-iron ore in the vicinity, I desired to find some way of making this coal available as a blast-furnace fuel; and my experiments proved to my satisfaction that a mixture of it in suitable proportion with the smalls of bituminous coking-coal (preferably after washing both coals), could be successfully treated in coke-ovens, the anthracite being absorbed or dissolved, so to speak, so as to produce a coke of uniform structure. The percentage of ash depended, of course, on the constituents of the original materials, and was reduced to a minimum by previous washing of these. The ability of the coke to carry burden in the furnace was not inferior to that of coke made from the same bituminous smalls without anthracite.

Controversies with the government, first of Hanover and afterwards of Prussia, concerning the mining concession of the coal (which I had acquired from the land-owners, but which the authorities refused to ratify), ultimately forced me to abandon this property, and with it the practicable development of my invention in the manufacture of anthracite-coke. Since coming to the United States, in 1867, I have been otherwise occupied, and have not pursued this matter further, except in private correspondence. I do not find in technical literature any references to it of as early date as my experiments above mentioned.

R. W. RAYMOND, New York City: I have received from Mr. Tooke Straker, Port Oram, N. J., an account of some experi-

ments, on a small scale, in the coking of anthracite culm mixed with bituminous slack. Mr. Straker reports, in substance, that while the bituminous coal alone gave a good, ordinary coke, and the culm alone gave no sign of coking, he obtained from a mixture of equal proportions fair coke, and from ¾ culm and ¼ bituminous coal a silvery, hard coke. For the benefit of members desiring to know what has been done in this direction, I will say, that the subject was brought up in September, 1875, at the Manchester meeting of the Iron and Steel Institute, in a paper by Mr. William Hackney, on "The Manufacture of Anthracite Coke in Wales," which was discussed by Mr. Riley, Professor Williamson, Dr. C. W. Siemens, and others (see *Jour. I. and S. Inst.*, 1875, part ii., p. 523). A summary of the Welsh process will be found in vol. L. of the Penn. Geol. Survey, p. 85, in the report of Mr. F. Platt. Vol. MM. of the same series contains a later report from Mr. Platt, giving the results of careful experiments, which included coking with and without the addition of pitch, and with varying proportions of anthracite and bituminous coal. Analyses of the materials are given, as well as tests of the product; and the conclusions reached by Mr. Platt are: 1. That a fuel can be made by coking mixed anthracite and bituminous slack which will answer the purposes of ordinary coke; 2. That the use of pitch has little, if any, advantageous effect; 3. That such coke can be best made in long, narrow ovens; 4. That about half-and-half is the best mixture to give strong coke. The best coke made in these experiments had 9.985 per cent. ash, and, tested in the foundry of the Cambria Iron Company, showed a lower smelting-capacity than Connellsville coke, besides requiring more time to do the same work, and crumbling badly. On the other hand, the original materials had not been washed before mixing and coking.

THE SECRETARY presented the paper of Mr. Blauvelt, on the Utilization of Anthracite Waste by Gasification in Producers, as a contribution to this discussion. (For this paper see page 625 of the present volume.)

JEROME L. BOYER, Reading, Pa. (Communication to the Secretary): We are using for our electric-light plant a mixture of 100 pounds bituminous coal with 400 pounds coal-dirt from the coal-yards in this city. The latter ranges in size from ¼-inch down to dust. We evaporate on a grate-surface of 36 square feet for each of our eight boilers about 7½ pounds of water per pound of fuel, and we think our fuel-mixture an economical one.

AXEL SAHLIN, New York City (Communication to the Secretary):

About a year ago, I made a test, to determine the economy of substituting bituminous coal as fuel, for crude oil or anthracite screenings (below the size of pea-coal). In determining such a matter, the size of the plant is of importance, and also the question, whether it is to be run continuously, or in the day-time only. There are also numerous local conditions to be considered. The plant in this case was required to evaporate, per hour, 36,000 pounds of water to steam of 80-pounds pressure. It comprised a number of tubular boilers 72 inches in diameter, 20 feet long, with fifty-six 4-inch steel tubes, furnishing a heating-surface of 1388 square feet per boiler. The shell was made of $\frac{3}{8}$-inch steel; the heads, of $\frac{1}{2}$-inch steel, were well braced. The boilers were fitted with McClave grates, for hard-coal firing, and ordinary double grate-bars for burning soft coal; and, where oil was utilized, the grate was covered with a double layer of fire-brick, and a checker-work of fire-brick was arranged across the grate, at a distance of about 2 feet 6 inches from the point of the burner. The grate-area was 30 square feet. The pipe connecting the boiler with the smoke-stack was brick-lined, and 29 inches diameter in the clear. The boilers were coupled in pairs to brick-lined iron smoke-stacks, 95 feet high, and 47 inches diameter in the clear. Unfortunately, no means were available for determining the temperature of the escaping gases in the stacks. Feed-water was supplied to the boilers at from 68° to 70° Fahr. In computing the tests, the temperature of the feed-water is uniformly assumed to have been 212° Fahr., and corresponding corrections were in each case made. The test lasted ten hours. This may seem a very short time, but as so many contributing factors had to be purely assumed, it was considered long enough for arriving at a conclusion sufficiently correct.

When firing coal, the grates were thoroughly cleaned once during the ten hours of the test, precisely as would have been the case in ordinary practice. Crude Lima oil was supplied under a uniform pressure of 20 pounds per square inch, and enough oil was fed to fill the combustion-chamber without unduly straining or crowding the boiler. The prices of fuel were assumed to be: oil, 2.06 cents per gallon; bituminous coal, $2.65 per ton; anthracite screenings (all below pea), $1.25 per ton, all delivered in the boiler-house. The test gave the following figures, which were used as the basis for estimates of economy:

Performance of 72 inches by 240 inches Tubular Boiler.

Fuel.	Crude oil.	Bituminous coal.	Anthracite screenings.
Duration of test,	10 hours.	10 hours.	10 hours.
Amount of fuel burned,	4746 lbs.	7120 lbs.	4723 lbs.
Value of fuel burned,	$13.41	$9.26	$2 95
Total water evaporated,	60,913 lbs.	47,206 lbs.	22,562 lbs.
Water evaporated per square foot of heating-surface per hour,	4388 lbs.	3400 lbs.	1625 lbs.
Actual average steam-pressure,	67 lbs.	65 lbs.	66 lbs.
Average temperature of feed-water,	68 Fahr.	68 Fahr.	70 Fahr.
Equivalent amount of water evaporated from 212° Fah. to 80 lbs. steam per lb. of fuel,	15.15 lbs.	7.87 lbs.	5.53 lbs.
Cost of fuel per one lb. water evaporated,	0.0220 cts.	0.0196 cts.	0.0131 cts.
Cost of fuel for evaporating 360,000 lbs. of water to 80 lbs. steam,	$79 20	$69.56	$47.16

As a first conclusion to be drawn from these figures, it was shown that, to evaporate the 360,000 pounds of water required, it would be necessary to operate: with oil, 6 boilers; with bituminous coal, 8 boilers; with anthracite screenings, 17 boilers.

The number of men and cost of labor for operating this number of boilers during ten hours would be:

			Per day.
For oil, 2 water-tenders,	$1.80		$3.60
For bituminous coal, 2 water-tenders,	1.80 =	$3.60	9.80
" " 4 firemen,	1.55 =	6.20	
For anthracite screenings, 4 water-tenders,	1.80 =	7.20	21.15
" " 9 firemen,	1.55 =	13.95	

Making allowances for depreciation, interest, insurance, etc., in proportion to the size of the plant actually in use, the following table was constructed:

Comparison of the Economical Results Obtained from the Use of Different Fuels in a Boiler-Plant, with an Evaporative Capacity of about 36,000 Pounds of Water per Hour during Ten Hours.

Fuel.	Crude oil.	Bituminous coal.	Anthracite coal.
Number of boilers in operation,	6	8	17
Actual value of fuel consumed,	$79.20	$69.56	$47.16
Cost of labor,	3.60	9.80	21.15
Depreciation, interest, insurance on plant at 15 per ct.,	9.00	12.00	25.00
Repairs, including grate-bars (assumed),	1.00	2.00	4.00
Total cost per day,	92.80	93.36	97.71

Besides the slight advantage which the above table indicates for the use of oil, there are others, which cannot be readily put in dollars and cents; the principal being, that the moment the steam is not required, the oil can be shut off, and all waste of fuel stopped. In burning coal, steam is generated for at least two hours after the machinery is stopped for the night, and the fuel on the grates is burnt for nothing. In spite of these advantages, it was concluded, that there would not be enough saved to justify the construction of tanks for the storage of the oil, the installation of pumps, distributing-pipes, etc.; and bituminous coal was settled upon as practically the best fuel for the plant in question.

[SECRETARY'S NOTE.—In the foregoing discussion, the word "culm" is used in its American sense, of anthracite waste, consisting of fine coal, coal-dust and dirt. In English usage, the term is sometimes applied to all anthracite. This explanation is given for the benefit of foreign readers.—R. W. R.]

THE UTILIZATION OF ANTHRACITE WASTE BY GASIFICATION IN PRODUCERS.

BY W. H. BLAUVELT, PHILADELPHIA, PA.

(Glen Summit Meeting, October, 1891.)

THAT the problem of utilizing anthracite waste is worthy of the thought bestowed upon it, may be shown in few words. In the year 1889, for instance, 35,408,000 tons of coal were sent to market from the anthracite regions of Pennsylvania. According to the report of the State Geological Survey, the amount of coal thrown on the dumps as waste, varies, with circumstances, from 44 to 60 per cent. of that shipped to market. The lower of these figures represents 15,579,000 tons of waste for that year, and it is obvious that if this, or any considerable portion of it, could have been saved, it would have constituted an important addition to the fuel-supply of the country. It is not surprising, therefore, that so much attention has been given to the utilization of this waste.

Among the many lines of experiment pursued with this object in view, the conversion of the carbon of the culm into producer-gas seems at first to promise the most satisfactory solution of the problem. It is well known that in foreign producer-plants almost all kinds of

combustible waste are converted into gas. Especially on the continent of Europe it is not uncommon to find gas made from turf or peat containing a large percentage of water, or from wet saw-dust, shavings, or green slabs just from the saw-mill, and successfully applied to high temperature work.

Our costly labor and cheap coal hardly warrant (except in peculiar cases) our going to such sources for fuel; but the demonstrated practicability of so doing makes it natural to infer that the finely-divided and impure state of anthracite culm would not prevent it from being a suitable material for making gas, while its higher percentage of carbon would, in the absence of special technical disadvantages, make it economically superior.

In the old days of the natural-draft producers, only the large sizes of anthracite were used, and the fuel-bed was kept of a proper thickness to insure an even distribution of the air-currents. When a forced blast came into use, the general opinion was that the problem of burning fine anthracite had been solved, and that it would be necessary only to increase the pressure of the blast in proportion to the fineness of the coal. Advocates of this method pointed to the blast-furnace as a proof of the correctness of their theory, forgetting that the conditions of working were quite different in the two cases, and that the successful use of blast at high pressure in the former by no means proved its practicability in the latter.

Some time ago the writer was interested in a series of experiments, extending over several months, with the Taylor gas-producer, which were conducted with the view both of perfecting the details of the apparatus, and of determining the suitability of various fuels for making producer-gas, and the best treatment of them in the producer.

The Taylor producer was chosen for this latter work, both on account of its being free from the defects of most producers, and because its design seemed to be specially adapted to the gasification of fine anthracite. A solid circular plate taking the place of the ordinary grate, and the fuel being supported on a deep bed of ashes, the fire can be cleaned with equal ease, without regard to the fineness of the fuel.

But the insurmountable difficulty is the action of the fine particles of fuel in obstructing the passage of the air through the fuel-bed. In the gasification of bituminous coal, the size of the coal makes little or no difference, as its coking keeps the fuel-bed open. But in the case of anthracite and other non-coking coals, the degree of permis-

sible fineness is limited by the tendency of the air-blast to choose the easiest line of travel; and when the fineness of the coal is increased beyond a certain point, this line will invariably be next the walls of the producer. The result is, that while the main body of fuel remains comparatively dead, combustion next the walls is very active, and the passage of the gases is so rapid that the carbonic acid formed by the first combustion of the coal has no time to be reduced to carbonic oxide. Thus, the gas produced is both small in amount and very poor in quality. Moreover, serious clinkers are formed by the high temperature, and the producer soon becomes choked.

By reducing both the depth of the fuel and the pressure of the blast, this irregular distribution of the air can be corrected; but to make this remedy effective when the finest anthracite is used, the rate of combustion must be reduced to a point where the corresponding increase in the size of the plant required makes its cost prohibitory.

Experience shows that the sizes known to the trade as "No. 1 buckwheat" and "pea" are best suited for use in a producer with forced blast. For in these sizes, the proportion between the interstices and the solid particles is such that a fuel-bed of sufficient height to make a gas low in carbonic acid can be carried, without causing irregularities in combustion, under a blast-pressure sufficient to insure a satisfactory rate of gas-making.

Especially in the older culm-heaps, there are considerable quantities of buckwheat or even larger-sized coal which could, doubtless, be recovered by washing and screening; but the profitableness of this is doubtful, and depends upon local circumstances.

Fine anthracite can be used in considerable quantities in a producer, when it is mixed with sufficient coking bituminous coal to keep the fuel-bed open; but it is usually cheaper to use buckwheat or pea alone, the excessive cost of the soft coal in the anthracite-regions more than counterbalancing the low price of the dust. Moreover, the mixture works only indifferently well.

Where freight-charges constitute an important proportion of the cost of the fuels, the large percentage of ash in the culm would make its economy very doubtful in comparison with well-prepared buckwheat, even were no difficulties encountered in its gasification. A series of experiments, conducted by the writer, in firing boilers with the two kinds of fuel, clearly showed this to be the case where the two sizes were used on grates capable of handling both equally well.

Both fuels being used under the same boiler, and the water evap-

orated being carefully weighed, the opportunity for comparison was a satisfactory one, and the conclusions were definitely in favor of the economy of the larger-sized and purer fuel.

Although invention has successfully overcome the difficulties of gasifying the smaller sizes of prepared anthracite, and such gas is used in a variety of industries in both high and low temperature work with entire satisfaction, it seems clear that we are as far as ever from the utilization of the refuse in this way. The most that can be done is to reduce the amount of refuse as far as possible by a more thorough separation of the buckwheat from the culm; looking to other processes for the final utilization of the remaining carbon.

THE USE OF THE McCLAVE GRATE AND ARGAND STEAM-BLOWER IN UTILIZING SMALL SIZES OF ANTHRACITE, OR BITUMINOUS SLACK IN BOILER- AND SIMILAR FURNACES.

BY RUFUS J. FOSTER, SCRANTON, PA.

(Glen Summit Meeting, October, 1891.)

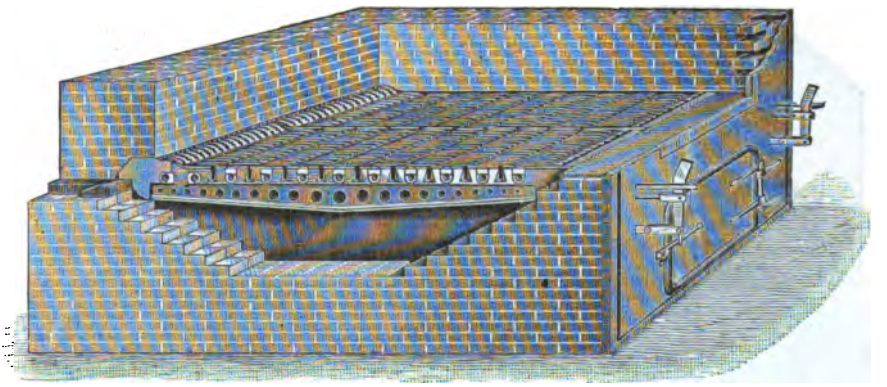
DURING the discussion on the Preparation and Utilization of Small Sizes of Anthracite (page 613 of present volume), several inquiries were made concerning the use of the McClave grate, which was mentioned by Mr. Coxe as having proved satisfactory in the generation of steam at his collieries, and also at a large number of industrial establishments in Scranton and elsewhere. For a more satisfactory reply to such inquiries than could be furnished in the oral discussion, I have prepared the following paper to be included in the published proceedings. And, as the use of the fuel under consideration necessarily requires the employment of a blower to change the fire, from a condition giving an incandescent radiant heat only to one giving a hot gaseous flame as well, I shall add a description of the McClave Argand blower and its method of action. The grate and blower combined form a complete and effective system for burning small fuels, such as anthracite culm, birdseye, buckwheat, bituminous slack, etc., in boiler and other furnaces; while the grate alone is now extensively used with the larger sizes of hard and soft coal in furnaces where blast is not required.

Fig. 1 is a view of the McClave grate as built in two sections. It is constructed on the principle best approved by practice to prevent warping and burning out; that is, it presents no long lines of

metal in contact with the fire. This is further illustrated in Fig. 2, which shows one section of the grate, with one journal-bar removed. This feature it possesses in common with some other grates, but it is, so far as I am aware, the only boiler-furnace grate that operates on the "pocket" principle: *i. e.*, when the grate-bars are thrown wide open, they form a series of pockets to receive the clinkers and ashes, which cannot pass through into the ash-pit until the bars have been thrown back into their normal position. A certain quantity of clinkers and ashes is thus, at each cut, instantly and uniformly removed from the underside of the fire. The cut-off movement is illustrated in Fig. 3.

The grate has also a thorough shaking-movement, during which

FIG. 1.



McClave Grate Complete in two Sections.

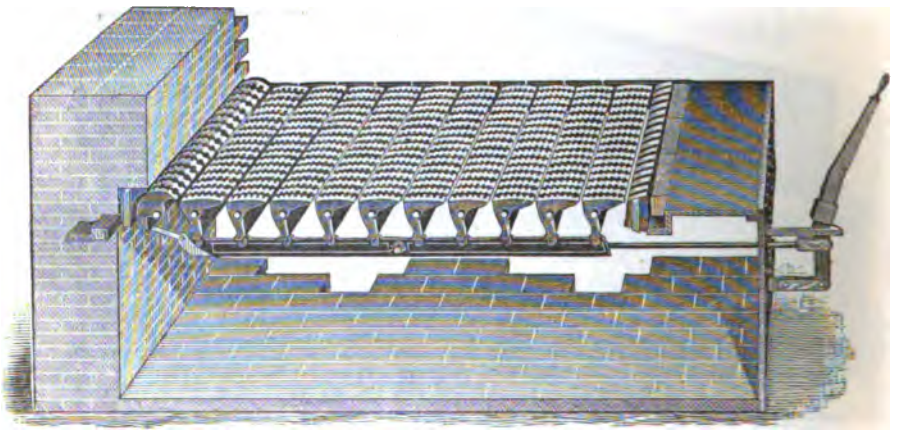
the front ends of the fingers of each bar remain equidistant from the curved back of the bar in front of it, so that there is at no time an increase in the size of the openings. This movement is well adapted to the breaking up of a soft-coal fire when it cakes, since nothing passes through into the ash-pit except the fine ash; nor is it necessary to open the fire-doors during the operation. The shaking-movement (illustrated in Fig. 4) is produced by vibrating the lever to and fro between the stop and the boiler-front, while the cut-off movement is accomplished by pulling the lever forward to its full extent and pushing it back again to the stop, in all cases leaving the stub-lever resting against the stop at the finish, as this indicates when the grate-bars are in their normal position. When this grate is used in connection with blast furnished by the Argand steam-blower, which keeps the clinkers soft and porous, a fine-fuel fire of 30 square

feet can be cleaned by this movement in 30 seconds, with the fire-doors shut.

Large grates are made up of two or more rows of bars, each row or section of grate having separate connections and stub-lever, and being operated by itself. One socket-handle serves to operate all the stub-levers successively, and two or three full movements of each row or section are quite sufficient to clean a heavy fire.

. The general practice in stoking small anthracite fuels, where these appliances are used, is to spread the fuel evenly over the entire surface of the fire at the same time, in thickness just sufficient to fairly cover the glowing surface. This is repeated as often as necessary, until the bed has accumulated to a depth of at least ten inches before

FIG. 2.



McClave Grate,—one Section, with one Journal-Bar removed.

cutting it down; and when it is cleaned it is not cut down until there is scarcely any fuel left, but the ashes and clinkers are simply cut away from the bottom of the bed, always leaving from 4 to 5 inches of fuel on the grate. After the fire is cleaned the surface is leveled with a rake or hoe, and any soft clinkers or boulders which may appear through the surface are broken down before putting on fresh fuel. The steam carried in the blast generally produces boulders or lumps, which are often mistaken for clinkers, but are, in reality, only lumps of fuel matted by the steam, and, if broken down and left in the fire, will be nearly all consumed. By following this method, a fire may be cleaned and fresh fuel may be put on and ignited so quickly that the steam will be kept at almost its maximum pressure during the process.

In all cases where anthracite culm is used as fuel it should be sprinkled with water before putting it on the fire, not enough to make it sloppy and heavy, but just enough to make the dust adhere to the particles of buckwheat. Anthracite screenings from the coal-yards should be treated in the same manner, and if they have lain out in the weather for a long time, it will be found advantageous to mix them with about one-fifth of their bulk of bituminous slack where this is available. Many have tried to burn this mixture with other appliances, but in most cases without success; with this blast, however, it makes an excellent fuel.

Bituminous slack alone is likewise spread as evenly as possible when throwing on fresh fuel, but is put on dry; and where the blast

FIG 3.

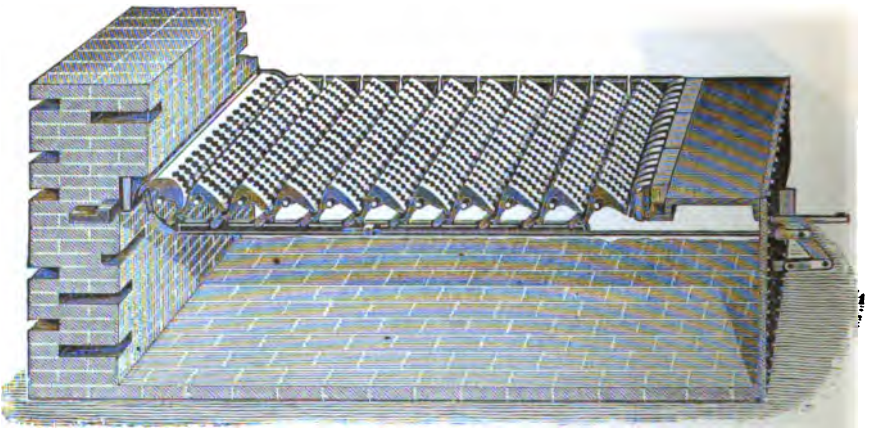
McClave Grate—Cut-off Movement.

is used the blower is shut off until the volatile matter has been liberated; then the blower is turned on again. In this case the shaking-movement of the grate is used frequently to break up the fire when it cakes; and it is sometimes necessary to break up the top crust and level the fire with a rake or hoe. In burning some kinds of slack, the cut-off movement is never used; but for soft-coal slack that forms clinkers, it has to be used when the fire gets so thick that the blast will not go through it properly. Where the blower is used a fire of slack may be allowed to accumulate until it is from 12 to 14 inches thick before it is cleaned.

The Argand blower is shown in Figs. 5 and 6, which will explain themselves to those acquainted with other steam-blowers. The

special merits of the Argand blower, as claimed by many who are using it are as follows: In the first place, it furnishes a large volume of air with a small amount of steam; again, the air and steam are thoroughly mixed in the shell or case of the blower before the blast is delivered into the ash-pit, and it makes but very little noise during operation, a feature fully appreciated by firemen. At a comparative test of six different kinds of steam-jet blowers recently made at Pine Brook colliery, Scranton, Pa., it was shown that the Argand blower inducted with the same quantity of steam, nearly twice as many cubic feet of air per minute, as any other blower represented in the trial, and more than twice as many cubic feet as some of its competitors.

FIG. 4.



McClave Grate—Shaking Movement.

It is now conceded by nearly all mechanical engineers who have given the subject special attention, that a blast furnished by under-grate blowers is better adapted to burn small fuels, such as buckwheat, birdseye, culm, slack, etc., than either a strong natural draught, or a draught produced by a jet or jets in the stack. Both of the latter methods so relieve the pressure on the upper surface of the fire, that the unconsumed gases escape into the stack before they have time to ignite; and as a boiler, in common with many other things, can be heated more effectively by a hot gaseous flame enveloping its entire heating-surface, than by an intense, radiant heat, emitted by incandescent carbon against a small portion of that surface only, this defect in the methods named is readily seen. Many resort to the use of an under-grate fan-blast as a remedy, since in this way the gases can be held back by the damper until they have

time to ignite and deliver up their heat-value, provided always that there is plenty of oxygen present to support their combustion. This method is, in my opinion, a step in the right direction, and towards economy in combustion for the purpose under consideration. But if it is taken into account that when a fan-blast is strong enough to furnish oxygen in sufficient quantity at the top of the fire to burn even a portion of the carbonic oxide produced by conversion, a vast quantity of nitrogen has also passed through the fire, which acts as a diluent, or in other words, an absorbent of heat, and that only a portion of the oxide formed in the fire can be reached by the oxygen in time to burn it; then the apparent gain by this method is materially lessened.

FIG. 5. FIG. 6.

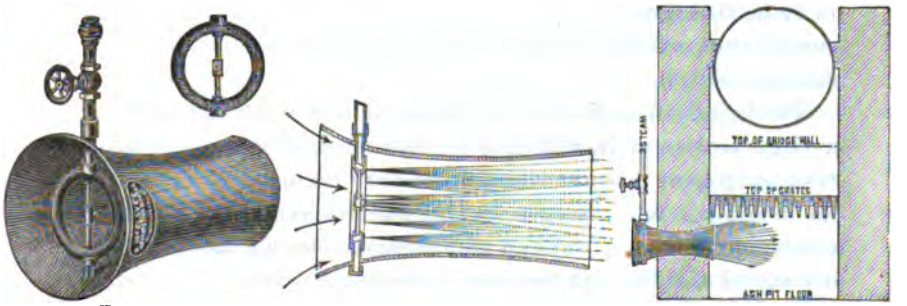


FIG. 5.—Argand Steam-Blower.
FIG. 6.—Section of Furnace, showing Operation of Argand Steam-Blower.

The device, therefore, of a combined air- and steam-blast, has gradually grown into favor, on account of the chemical and mechanical effect of the steam on the fire; and many different kinds of steam-jet blowers have from time to time been placed on the market for this purpose. It was soon discovered, however, that the majority of these blowers furnished by far too much steam; for it is a well-established fact, that while a small quantity of steam is a valuable constituent in the blast, an excess of steam defeats the very purpose for which it was intended; first, by furnishing an insufficient supply of air, with carbonic oxide as a resultant, the heat evolved being about one-third of that obtained by complete combustion; second, by overtaxing the decomposing power of the fire with too large a quantity of steam, which passes through the fire simply as steam, thereby lowering the temperature of the gases above the fire in many cases below the point of ignition, or, if the temperature is not reduced to that degree, a volume of vaporous flame of low temperature is the

result, due to the mechanical mixing of the steam with the burning gas as it passes back under the boiler; nearly the entire product of the fire-bed in such a case being carbonic oxide, with but a small quantity of available oxygen present to combine with it above the fire.

It is claimed, however, that the chemical effect of a moderate quantity of steam is to furnish oxygen by dissociation at the proper point or stratum in the fire, to partially prevent the production of carbonic oxide, by conversion from carbonic acid gas in its passage through the glowing carbon in the upper zone of the fire; and also to help consume the carbonic oxide that will be formed; for it should be borne in mind that heat is always absorbed in the production of carbonic oxide by conversion (reduction); and it should also be kept in view, that the oxide formed in this manner must pass off unconsumed, thus wasting its heat-value, unless oxygen is furnished to combine with it.

The hydrogen set free by the dissociation of the steam, or at least a large portion of it, will also be consumed by coming in contact with oxygen at a lower temperature than the point of dissociation, for in addition to the oxygen that has passed through the fire without combining, there is always an admission of some air around the fire-doors, and also through the ventilators in the doors.

What is required, therefore, for boiler-furnaces, and other furnaces of a similar nature, is neither a radiant heat under a small portion of the heating surface, nor a draught produced by a jet or jets in the stack, nor yet a forced draught under the grate that will turn the fire into a gas-producer pure and simple, which gives only one-third of the heat-value of the fuel; but rather a happy medium between extremes, that will produce a sufficient volume of gas, and furnish oxygen at the proper point to burn it; getting thereby, not only better combustion, but also flame enough at a high temperature to envelop the entire heating-surface of the boiler, maintaining at the same time a high temperature in the bed of the fuel, thereby securing an intense radiant heat as well.

The mechanical effect of the steam is that it keeps the clinkers soft and porous, so that the blast will readily pass up through the entire bed of fuel uniformly instead of being forced to pass between solid clinkers wherever it can find an opening, thus producing what are generally termed "forge-flames" under the boiler. This is usually the case with a fan-blast, for an all air blast tends to form the clinkers into compact slabs through which the air cannot pass.

Another mechanical effect of the steam is that it moistens the fine ashes from the grate upward as fast as they are made. This serves to knit them together and hold them in the strata in which they were produced instead of permitting them to be blown up into the burning surface to choke it by filling up the interstices between the particles of fuel.

While fully aware of the fact that heat is absorbed in the decomposition of steam, yet I think the practical results obtained by the use of this blower warrant the conclusion that the heat absorbed is far more than compensated for by the beneficial results obtained, and that a properly constructed steam-jet, under-grate blower that will induct a large volume of air with the use of a small amount of steam will, everything considered, furnish a better blast for the purpose indicated than any other means that has yet been employed.

A simple yet very important feature of successful practice in burning fine fuels, where the Argand blowers are used to furnish blast, is to close the damper in the chimney or stack just so that the burning gases will not blow out through the fire doorway when the door is opened. As already observed, where there is a strong chimney-draught in connection with under-grate blowers, a large percentage of the gases escapes without igniting. The damper should, therefore, be so regulated that the largest possible volume of gaseous flame may be produced in the furnace. Where the chimney-draught is weak it may be necessary to keep the damper wide open, but in the majority of cases it will be found not only beneficial, but absolutely essential to close the damper to the extent described in order to obtain the best results.

Discussion.

R. W. Raymond, New York City: Mr. Foster's statement of the chemical and mechanical action of steam in an under-grate blast seems to me liable to misunderstanding, although it does not, when properly construed, involve, so far as I can see, any erroneous theory. A brief additional summary of the facts and principles involved is offered, therefore, rather in confirmation than in contradiction of the main argument, with which I fully agree. But the extent of popular error concerning the calorific relations of the steam-jet in such appliances seems to call for very careful statements of the theory.

1. No possible use of steam can cause a gain of heat. If steam be introduced into a bed of incandescent carbon it is decomposed

into hydrogen and oxygen. This is not dissociation; and the temperature of dissociation, *i. e.*, the temperature at which hydrogen and oxygen would separate without the presence of carbon, or would, if separate, exist side by side without uniting, is not produced in ordinary combustion without the use of regenerators.

2. The heat absorbed by the reduction of one pound of steam to hydrogen is much greater in amount than the heat generated by the union of the oxygen thus set free with carbon, forming either carbonic oxide or carbonic acid. Consequently, the effect of steam alone upon a bed of incandescent fuel is to chill it, as all water-gas experts are aware. In every water-gas apparatus, designed to produce by means of the decomposition of steam a fuel-gas relatively free from nitrogen, the loss of heat in the producer must be compensated by some reheating device.

3. This loss may be recovered if the hydrogen of the steam is subsequently burned, to form steam again. Such a combustion of the hydrogen is contemplated, in the case of fuel-gas, as secured in the subsequent use of that gas. In boiler-furnaces, such as we are now considering, it must be secured, if at all, before the flame escapes from contact with the boiler; and in the steam-jet undergrate blowers it is sought to secure the object by supplying with the steam air enough to furnish surplus oxygen on the upper side of the fuel-bed, at which point both carbonic oxide and hydrogen are to be oxidized. Assuming this oxidation to be complete as regards the hydrogen, the use of steam will thus have caused neither gain nor loss of heat, but a simple transference, the heat absorbed by steam-decomposition on the lower side of the fire being restored by hydrogen-combustion on the upper side. In practice, it may be doubted whether this restoration is ever complete. But it is certain that an excess of steam would defeat the reaction altogether, and that there must be a certain proportion of steam, which permits the realization of important advantages, without too great a net loss in heat.

4. The advantage to be secured consists, in my judgment, principally in the transfer of heat from the lower side of the fire, where it is not wanted, to the upper side, where it is wanted. The decomposition of the steam below cools the fuel and the grate-bars, whereas a blast of air alone would produce, at that point, intense combustion (forming at first CO_2), to the injury of the grate, the fusion of part of the fuel, etc.

5. The proportion of steam most economical is not easily determined. The temperature of the steam itself, the nature of the fuel-

mixture, and the use or non-use of auxiliary air-supply, introduced, as in the Boëtius and other furnaces, into the gases above or beyond the fire-bed, are factors affecting the problem. But in the case before us the desideratum has not been to realize the utmost theoretical calorific effect from fine fuels, but to burn them with practical success at all. I do not understand it to be claimed for any grate or blower that it will get as much out of these fine fuels as out of an equal weight of coarser sizes. And the combined apparatus described by Mr. Foster appears to be an excellent solution of the general problem, calculated to operate smoothly in everyday practice, and to secure as high a degree of effective economy as could be reasonably expected under the circumstances.

CENTRIFUGAL VENTILATORS.

BY R. VAN A. NORRIS, WILKES-BARRE, PA.

(Glen Summit Meeting, October, 1891.)

ALTHOUGH mechanical appliances for the ventilation of mines have been known since very early times (one being mentioned in Agricola's *De Re Metallica*, 1657), it is only within the last forty years that such appliances have been generally used. A great number of different ventilators have been invented and used, but the various forms of centrifugal machines having proved their efficiency, have now almost completely replaced all others. Most if not all of the machines in use in this country are of this class, being either open-periphery fans, or closed, with chimney and spiral casing, of a more or less modified Guibal type. The theory of such machines has been demonstrated by Mr. Daniel Murgue in *Theories and Practice of Centrifugal Ventilating Machines*, translated by A. L. Steavenson. His statement may be condensed thus:

Let a = orifice in a thin plate, of such area that its resistance to the passage of a given quantity of air equals the resistance of the mine.

o = orifice in a thin plate of such area that its resistance to the passage of a given quantity of air equals that of the machine.

Q = quantity of air passing in cubic feet per minute.

V = velocity of air passing through a.

V_0 = velocity of air passing through o.

h = head in feet air-column to produce velocity V.

h_0 = head in feet air-column to produce velocity V_0.

$$Q = 0.65 a V. \qquad V = \sqrt{2gh}.$$

$$Q = 0.65 a \sqrt{2gh}, \quad . \quad . \quad . \quad (1).$$

$$a = \frac{Q}{0.65\sqrt{2gh}} = \text{equivalent orifice of mine,}$$

or, reducing to water-gauge in inches and quantity in thousands of feet per minute:

$$a = \frac{.403\, Q}{\sqrt{W.G.}}.$$

$$Q = 0.65\, o\, V_0. \qquad V_0 = \sqrt{2gh_0}.$$

$$Q = 0.65 o \sqrt{2gh_0}, \quad . \quad . \quad . \quad (2).$$

$$o = \sqrt{\frac{Q^2}{0.65^2 h_0\, 2g}} = \text{equivalent orifice of machine.}$$

The total theoretical depression H, which any centrifugal ventilator can produce, is the algebraic sum of the following depressions caused by its motion:

1st. The depression, h_1, due to the speed, V_1, to which the motionless air is brought and with which it traverses the fan-intake,

$$h_1 = -\frac{V_1^2}{2g}, \quad . \quad . \quad . \quad (3).$$

2d. The depression, h_2, due to the slowing of the air from its entrance to its exit in the expanding spaces between the blades, entering at a speed, V_2, and leaving at a speed, V_3,

$$h_2 = \frac{V_2^2}{2g} - \frac{V_3^2}{2g}, \quad . \quad . \quad . \quad (4).$$

But the speed of entry, V_2, is the resultant of V_1 and a speed equal and opposite to the tangential speed t of the inner edges of the blades whence $V_2^2 = V_1^2 + t^2$ and

$$h_2 = \frac{V_1^2}{2g} + \frac{t^2}{2g} - \frac{V_3^2}{2g}. \quad . \quad . \quad . \quad (5).$$

3d. The depression h_3, due to centrifugal force,

$$h_3 = \frac{T^2}{2g} - \frac{t^2}{2g}, \quad . \quad . \quad . \quad (6).$$

where T = tangential speed of periphery, and

t = tangential speed of inner edges of blades.

This value of h_3 may be deduced as follows:

Let x = distance from center of fan of a prismatic volume of air between two blades.

dx = height of prismatic column.

S = base of column.
D = density.
ω = angular velocity of rotation.
r = radius of inside edges of blades.
R = radius of tips of blades.
dF = centrifugal force of rotation.

$$\text{Mass} = \frac{SdxD}{g}, \qquad dF = \frac{SdxD}{g}\omega^2 x$$

Increase in depression from one surface to another is obtained by dividing by S, and dividing again by D to get depression in feet of air-column.

$$dh_3 = \frac{\omega^2 x dx}{g},$$

integrating between R and r:

$$\int dh_3 = \int_r^R \frac{\omega^2 x dx}{g}.$$

$$h_3 = \frac{\omega^2 R^2}{2g} - \frac{\omega^2 r^2}{2g} = \frac{T^2}{2g} - \frac{t^2}{2g}, \quad . \quad . \quad (6).$$

4th. The depression h_4 due to the expanding chimney:

V_4 = speed of air at bottom.
V_5 = speed of air at top.

$$h_4 = \frac{V_4^2}{2g} - \frac{V_5^2}{2g}, \quad . \quad . \quad . \quad (7).$$

and as V_4 is the resultant of the tangential speed T and the speed V_3,

$$V_4^2 = T^2 + V_3^2 - 2TV_3\cos\alpha,$$

if α = angle at which tips of blades strike the circumference,

$$h_4 = \frac{T^2}{2g} + \frac{V_3^2}{2g} - \frac{TV_3\cos\alpha}{g} - \frac{V_5^2}{2g}, \quad . \quad (8).$$

and $$H = h_1 + h_2 + h_3 + h_4 = \frac{T^2}{g} - \frac{TV_3\cos\alpha}{g} - \frac{V_5^2}{2g}. \quad (9).$$

If we make the tips of the blades radial, $\alpha = 90°$, $\cos\alpha = 0$,

$$H = \frac{T^2}{g} - \frac{V_5^2}{2g},$$

and if the velocity at the top of the chimney is small enough to be neglected, or $V_5 = 0$,

$$H = \frac{T^2}{g}, \quad . \quad . \quad . \quad . \quad (10).$$

Hence: *The theoretical depression which can be produced by any centrifugal ventilator is double that due to its tangential speed.*

$$H_1 = h_1 + h_2 + h_3 = \frac{T^2}{2g} - \frac{V_3^2}{2g}$$

is the expression for the theoretical depression which can be produced by an uncovered ventilator; this reaches a maximum when the air leaves the blades without speed, that is, $v_3 = 0$.

$$H_1 = \frac{T^2}{2g}, \quad . \quad . \quad . \quad . \quad (11).$$

Hence: *The theoretical depression which can be produced by any uncovered ventilator is equal to the height due to its tangential speed, and one-half that which can be produced by a covered ventilator with expanding chimney.*

As the theoretical depression, H, can never be actually attained, we can substitute for it the depression H_2, letting C = coefficient of reduction, whence $H_2 = CH$; and as the effective depression, h, is this quantity less the depression h_0 absorbed in the machine itself, we have

$$CH = H_2 = h + h_0; \text{ and } h = H_2 - h_0.$$

From (1) and (2):

$$Q = .65\, a\sqrt{2gh},$$

$$Q = .65 o\sqrt{2gh_0},$$

we have

$$\frac{h_0}{h} = \frac{a^2}{o^2}, \quad . \quad . \quad .$$

$$h_0 = h\frac{a^2}{o^2}$$

$$h = H_2 - h\frac{a^2}{o^2}$$

$$Q = \frac{.65 a\sqrt{2gH_2}}{\sqrt{1 + \frac{a^2}{o^2}}} = \frac{.65 aT\sqrt{2C}}{\sqrt{1 + \frac{a^2}{o^2}}}, \quad . \quad (12).$$

$$h = \frac{H_2}{1 + \frac{a^2}{o^2}} = \frac{CT^2}{g\left(1 + \frac{a^2}{o^2}\right)}, \quad . \quad . \quad (13).$$

(12) and (13) being expressions for the yield and depression produced by any machine when its equivalent orifice, the equivalent orifice of the mine, and its coefficient of reduction C are known. From these it follows that so long as the condition of the mine remains constant:

The volume produced by any ventilator varies directly as the speed of rotation.

The depression produced by any ventilator varies as the square of the speed of rotation.

As the resistance decreases, a increases, and for the same tangential speed with decreased resistance the quantity of air increases and the depression diminishes.

An attempt has been made to test experimentally the effect, on the efficiency of fans in actual use, of modifications in the design of their various parts and in the condition of the mines; and through the courtesy of the managers, engineers, and superintendents of a number of coal companies, the results of tests of a number of fans shown in the accompanying table (pages 642 and 643), with diagrams of the fans giving them, have been obtained.

Fan No. 1, shown in Fig. 1, is in use for ventilating part of the No. 4 slope workings of the Susquehanna Coal Company at Nanticoke, Pa. This is in effect two fans, working side by side on different shafts. Each fan is 20 feet in diameter, having eight blades with curved iron inside to let the air enter at the proper angle, the tips of the blades, however, being curved backward until they are tangent to the circumference (instead of normal, as theory would seem to require). Each fan has two side-openings, 8 feet 10 inches in diameter, for the entrance of air, and a disk of sheet-iron 7 feet in diameter in the center. They are on separate shafts, each driven by a 24-inch 8-ply rubber belt on a 6-foot pulley. The engines are a pair of 20-inch by 36-inch, driving two 12-foot pulleys on the same shaft. The eccentricity of the spiral casing, as shown in the diagram, Fig. 1, is very great, being nearly one-half the diameter of the fan. The spiral begins slightly above the center at such a point that the space between two blades at least is always cut off to prevent re-entries of air, and from this point it is designed to allow each blade to deliver air continually and freely throughout its revolution, the entire body of air being swept along and increasing at each point in the spiral. Each fan is provided with a sliding shutter to regulate the area of the outlet.

The two fans working together were tested in April, 1891 (Experiments A, B, D, E in the table), at speeds varying from 84 to 123 revolutions. The power of the engines in each experiment was obtained from the average of twenty indicator cards, taken five from each end of each cylinder with two Tabor indicators, at intervals of about two minutes, while the air was being measured. The air-way

Table of Fan Experiments.

No. of Fan.	Experiment.	Owner and Location.	Type of Fan.	Diameter.	Width.	No. of Inlets.	Diam. Inlets.	Shutter?	Revolut'ns per Min., Engine.	Revolut'ns per Minute, Fan.	Periph'ry Speed, Feet per Min.	Cubic feet Air Per Minute.	Cubic feet Air Per Revolut'n.	Cubic'l contents of Fan Blades.	Cub. ft. Air per 100 ft. Periphery Motion.	Water Gauge, Inches.	Horse Power in Air.	Indicat'd Horse Power of Engine.	Efficiency Engine & Fan.	Equivalent Orifice of Mine, Sq. Feet.	Remarks.
1	A	Susquehanna Coal Co. Double Fan.	Guibal.	20	6	4	8'10"	Yes..	42	84	5517	236,684	2818	3040	4290	1.80	67.13	88.40	75.9	April, 80 sq.ft. Aug., 75	Fig. 1. Two fans on separate shafts, driven by pair 20" x 36" engines, belted to drive 2 to 1, with two 24" 8-ply rubber belts.
	B		Double.	.....	.....	...	.....	.....	50	100	6282	336,862	3369	3040	5393	2.50	132.70	155.43	85.4		
	C	.....	.....	.....	.....	...	.....	.....	50	100	6282	298,080	2981	3040	4745	2.60	121.94	143.42	85.02		
	D	.....	.....	.....	.....	...	.....	.....	55½	111	6973	347,396	3130	3040	5002	3.20	175.17	209.64	83.6		Experiments A, B, D, E, F, G, April, 1891.
	E	.....	.....	.....	.....	...	.....	.....	61½	123	7727	394,100	3204	3040	5100	3.60	223.56	295.21	75.7		Experiment C, August, 1891.
	F	.....	.....	Only left hand					50	100	6282	188,888	1889	1520	3007	1.40	41.67	97.99	42.5		Each fan has two inlets and sheet-iron disc.
	G	.....	.....	Fan running					65	130	8167	274,876	2114	1520	3366	2.00	86.63	194.95	44.6		
2	A	Susq. Coal Co. No. 6 Slope.	Guibal.	20	6	2	8'10"	No...	59	59	3702	59,587	1010	1520	1610	1.20	11.27	16.76	67.83	22	Same as One Fan, Fig. 1, except tips of blades radial.
	B	.....	.....	.....	.....	...	.....	.....	83	83	5208	82,969	1000	1520	1593	2.15	27.86	48.54	57.38		No shutter, no disc. Pair 16" x 24" engines.
3	A	Susq. Coal Co. Ross Seam. No 2 Shaft.	Guibal.	25	8	1	11'6"	Yes..	40	40	3140	49,611	1240	3096	1580	0.87	6.80	13.82	49.2	32	Fig. 2. Single opening. 24" x 32" engine.
	B								70	70	5495	137,760	1825	3096	2507	2.55	55.35	67.44	82.07		
4	A	Susq. Coal Co, Lee Seam.	Guibal.	25	8	1	11'6"	Yes..	62	62	4867	225,288	3634	3096	4650	1.45	51.44	62.93	81.74	75	Same as Fig. 2. Single inlet. Blades flat. 18"x36" eng.
5	A	Del. & Hudson Canal Co. Baltimore Tunnel.	Guibal.	17½	4	4	8	No...	28	1539		68,337	2440	1522	4440	0.10	1.07	.....	.....	.....	Fig. 3. From Mine Inspector's Report 1881: No indicator; fans run with throttle full open, and 75 per ct. of boiler pressure assumed as mean eff. pressure. 2 fans on same shaft. Pair 16x30 eng. 24" belt.
	B		Double.	.....	.....	...	.....	.....	50	2749		147,232	2944	1522	5356	0.50	11.60	28.55	40.63		
	C	.....	.....	.....	.....	...	.....	.....	69	3793		205,761	2982	1522	5451	1.00	32.42	45.98	70.50	83	
	D	.....	.....	.....	.....	...	.....	.....	96	5278		299,600	3121	1522	5676	2.15	101.50	120.64	84.10		
6	A	Union Coal Co. Pennsylvania Colliery.	?	18	6'4"	2	8	No...	80	80	4520	66,560	832	1293	1473	1.35	14.13	23.60	60.		Fig. 4. 14" x 24" engine.
	B								100	100	5650	92,160	922	1293	1631	2.10	30.50	43.37	70.	25	
	C	.....	.....	.....	.....	...	.....	.....	120	120	6780	109,440	912	1293	1467	2.90	50.00	81.03	61.		
7	A	Min'l. R.R. & Min. Co. Cameron Colliery.	Guibal.	18	4'11"	2	8	No...	56	110	6215	71,590	651	1004	1152	2.45	18.00	30.80	59.	18	Fig. 5. 18" x 36" engine.
8	A	Lehigh & W. B. Coal. Co. Stanton Old Fan.	Guibal.	35	11'8"	1	17'6"	No...	46	46	5060	228,850	4975	8419	4523	1.95	70.62	98.45	71.73	66	Fig. 6. 20" x 48" engine.
9	A	Lehigh & W. B. Coal Co. Stanton New Fan.	Guibal.	35	11'8"	1	17'6"	No...	46	46	5060	234,830	5105	8419	4641	1.85	68.46	94.60	72.37	68	Fig. 7. 20" x 48" engine.
	B								55	55	6050	277,380	5043	8419	4585	2.75	92.93	159.02	57.81		
10	A	Lehigh & W. B. Coal Co. Empire.	Guibal.	24	10	1	12	No...	66	66	4976	134,662	2040	3393	2686	1.95	41.38	62.70	66.00	44	Fig. 8. 18" x 36" engine.
	B								73	73	5504	194,200	2660	3393	3528	2.50	75.69	101.07	75.69		
11	A	Lehigh & W. B. Coal Co. S. Wilkes-Barre.	Capell..	12'6"	12'0	2	7'4"	No...	50	150	4887	177,280	1182	964	3628	1.35	37.79	90.10	41.80	64	Fig. 9. 24" x 36" engine.
	B			.....	.....	...	.....	.....	60	180	5865	215,200	1195	964	3669	1.65	56.00	123.00	45.30		
11½	A	Lehigh & W. B. Coal Co. Hollenback.	Guibal.	35	11'8"	1	17'6"	No...	45	45	4950	217,338	4830	8419	4390	2.25	77.05	84.4	91.30	58	Fig. 7. 20" x 48" engine. This efficiency is doubtful.
	B			.....	.....	...	.....	.....	50	50	5500	257,520	5150	8419	4682	2.40	97.40	131.3	74.20	66.5	
12	A	East Howle Colliery. Cumberland, England.	Capell..	12	10	2	7	No...	50	200	7540	133,198	666	746	1767	3.35	70.30	102.79	68.40	26.9	From Colliery Engineer, October, 1890.
	B			.....	.....	...	.....	.....	50	200	7540	156,510	782	746	2076	3.30	81.35	113.61	71.60	31.6	Pair engines, 20" x 36" One 24" belt. Driving 4 to 1.
	C	.....	.....	.....	.....	...	.....	.....	50	200	7540	180,809	904	746	2398	3.05	86.89	129.07	67.30	38.3	
	D	.....	.....	.....	.....	...	.....	.....	50	200	7540	209,150	1046	746	2774	2.80	92.50	150.08	61.70	46.3	

13	A	South-West Coal & Coke	Guibal.	25	8	1	12	No...	10	10	785	28,896	2890	3022	3680	0.10	0.45	1.80	35.		Fig. 10. 20″ x 36″ engine.
	B	Company.	.....	.....	.....	...	.....	.....	20	20	1570	57,120	2856	3022	3637	0.20	1.80	8.70	49.		
	C	Morewood Mine.	.....	.....	.....	...	.....	.....	25	25	1962	66,640	2665	3022	3399	0.29	2.90	6.10	48.		
	D	.....	.....	.....	.....	...	.....	.....	30	30	2355	73,080	2436	3022	3103	0.40	4.60	9.70	47.	52	
	E	.....	.....	.....	.....	...	.....	.....	35	35	2747	94,080	2688	3022	3425	0.50	7.40	15.00	48.		
	F	.....	.....	.....	.....	...	.....	.....	40	40	3140	112,000	2800	3022	3567	0.70	12.30	24.90	49.		
	G	.....	.....	.....	.....	...	.....	.....	50	50	3925	132,700	2654	3022	3381	0.90	18.80	38.80	48.		
	H	.....	.....	.....	.....	...	.....	.....	60	60	4710	173,600	2893	3022	3686	1.35	36.90	66.40	55.		
	I	.....	.....	.....	.....	...	.....	.....	70	70	5495	203,280	2904	3022	3718	1.80	57.70	107.10	54.		
	J	.....	.....	.....	.....	...	.....	.....	80	80	6280	222,320	2779	3022	3540	2.25	78.80	152.60	52.		
14	A	Lehigh Valley Coal Co.	Guibal.	35	12	1	17′6″	No...	48	48	5278	177,000	3687	8659	3353	2.25	62.70	82.20	76.2	47	Fig. 7. 24″ x 42″ engine.
		Dorrance Colliery.																			
15	A	L. V. C. Co. Henry Coll'ry	Guibal.	30	10	1	15′	No...	45	45	4241	245,000	5444	5302	5777	1.35	52.12	60.22	86.55	84	Fig. 11. 30″ x 48″ engine.
16	A	L. V. C. Co. Packer No. 5	?	20	6	1	12′	No...	58	58	3644	84,149	1451	1206	2309	0.95	12.60	23.04	54.35	35	Fig. 12. 17″ x 24″ engine.
	B	.....	.....	.....	.....	...	.....	.....	80	80	4994	117,573	1470	1206	2352	1.80	33.35	54.60	61.00		
17	A	Peasley Colliery, Lanca-	Guibal.	24	8	...	.....	Yes..	.....	70	5273	85,285	1218	2794	1617	2.44	32.74	40.00	80.18		From A. A. Atkinson in Colliery Engineer.
	B	shire, England.	.....	.....	.....	...	.....	.....	.....	80	6026	98,512	1231	2794	1628	3.00	46.56	61.12	76.17	22	Fan fitted with Walker shutter. 20″ x 20″ engine.
	C	.....	.....	.....	.....	...	.....	.....	.....	90	6780	107,300	1192	2794	1580	3.65	61.71	82.12	75.14		
18	A	Del. Lac. & Western RR.	Open ...	12	3′4	2	6′	.....	12½	26	979	17,640	498	283	1801	0.08	0.23	0.63	36.80		From Mine Inspector's Report, 1876.
	B	Avondale.	.....	.....	.....	...	.....	.....	25	52	1958	29,498	569	283	1506	0.25	1.16	2.33	49.87	25	Experiments made, 1872. "Richards" Indicator.
	C	.....	.....	.....	.....	...	.....	.....	50	104	3916	51,842	676	283	1824	0.70	5.72	12.36	46.26		14″ x 30″ engine.
19	A	D. L. & West'n RR. Co.	Open ...	16	4′6″	2	8′	.....	57	110.6	5558	132,282	1196	679	2380	1.80	38.09	58.92	64.00	40	Fig. 13. April, 1891.
	B	Bellevue.	.....	.....	.....	...	.....	.....	59	114.5	5758	160,714	1404	679	2794	1.60	40.57	58.71	69.00	60	" "
	C	.....	.....	.....	.....	...	.....	.....	57	110.6	5558	188,146	1701	679	3384	1.20	35.57	58.00	61.40	68	" "
	D	.....	.....	.....	.....	...	.....	.....	58	112.6	5658	152,188	1351	679	2688	1.90	45.56	68.15	66.8	44.1	Jan. 29, 1891.
	E	.....	Closed..	.....	.....	...	.....	Yes..	58	112.6	5658	142,159	1262	679	2512	1.60	35.84	51.52	69.5	45.2	Fig. 14. Feb. 2, 1891.
	F	.....	Closed..	.....	.....	...	.....	No...	58	112.6	5658	145,885	1296	679	2578	1.75	40.22	54.22	74.2	43.8	Fig. 15. Feb. 12, 1891.
	G	.....	Open ...	.....	.....	...	.....	.....	.....	112	5628	None...	.....	679	.....	1.35	.....	.....	0	0	Fig. 13. June 11, 1890.
	H	.....	Closed..	.....	.....	...	.....	Yes..	.....	112	5628	None...	.....	679	.....	2.30	.....	.....	0	0	Fig. 14. Sept. 29, 1890.
	I	.....	Closed..	.....	.....	...	.....	No...	.....	112	5628	None...	.....	679	.....	2.57	.....	.....	0	0	Fig. 15. Feb. 11, 1891.
	J	.....	Open ...	.....	.....	...	.....	.....	57	110.6	5558	136,387	1233	679	2453	1.20	25.78	.....	.....		Fig. 13. July 26, 1890.
	K	.....	Closed..	.....	.....	...	.....	No...	57	110.6	5558	154,738	1399	679	2783	1.60	39.01	.....	.....	49	Fig. 14. Oct. 1, 1890.
	L	.....	Closed..	.....	.....	...	.....	Yes..	57	110.6	5558	143,597	1298	679	2583	1.50	33.94	.....	.....		Fig. 14. Shutter. Dec. 23, 1890. Engine 16″ x 30″.
20	A	D. L. & West'n RR. Co.	Guibal.	25	8	2	12½′	No...	43	43	3377	135,600	3154	2945	4016	0.70	14.96	23.95	62.5	65	Fig. 16. Mine normal. 18″ x 36″ engine.
	B	Holden Fan.	.....	.....	.....	...	.....	.....	43	43	3377	113,140	2631	2945	3350	0.80	14.26	22.85	62.4	51	Mine choked.
	C	.....	.....	.....	.....	...	.....	.....	40	40	3142	194,510	4863	2945	6190	0.47	14.56	20.71	70.3	114	Mine doors open.
21	A	D. L. & West'n RR. Co.	Open ...	16	4′6	2	8′	.....	48	88.6	4452	177,855	2007	679	3995	0.85	23.82	43.97	54.2	77	Fig. 13. Mine normal. 18″ x 36″ engine.
	B	Dodge Fan.	.....	.....	.....	...	.....	.....	60	110.8	5568	201,135	1815	679	3612	1.10	34.86	70.49	49.5		" "
	C	.....	.....	.....	.....	...	.....	.....	48	88.6	4452	231,225	2610	679	5194	0.75	27.33	44.93	60.8	106	Mine doors open.
22	A	Coxe Bros. & Co.	Guibal.	20	6	1	7	Yes..	62	62	3890	51,084	824	1654	1313	1.00	8.05	19.82	40.62	20.4	Fig. 17. Engine 16″ x 24″.
		No. 2 Slope, Drifton.																			
23	A	Phila. & R. C. & Iron Co.	Guibal.	21	7′2″	1	10′6″	No...	70	70	4620	126,000	1800	1862	2727	0.85	16.8	73.00	23.	55	Fig. 18. Engine 14″ x 36″. NOTE.—Air measurem'ts
		Henry Clay Collery.																			taken in same manner as for Fans 19, 20 and 21.
24	A	Phila. & R. C. & Iron Co.	Guibal.	15	5′2″	1	7′6″	No...	100	100	4712	65,000	650	685	1167	1.60	16.4	68.00	24.1	20.6	Fig. 18. Engine 12″ x 30″.
		Mahanoy City Colliery.																			
25	A	Phila. & R. C. & Iron Co.	Guibal.	18	6′2″	1	9′	No...	80	80	4523	80,000	1000	1177	1770	2.75	34.67	58.00	59.8	19.3	Fig. 18. Engine 16″ x 18″.
		W. Bear Ridge Colliery.																			

NOTE.—The Air measurements in all these D. L. & W. R.R. Co. experiments were made with anemometer held stationary in center of airway, giving quantities of air much too high.

back of the fan was divided by fine wires into four portions, and a two-minute measurement of air was taken in each, the anemometer being moved backward and forward over the whole space of each portion to obtain, as nearly as possible, the true average velocity. The quantity of air passing was calculated from the average of the four measurements, making allowance in the area of the passage for the space occupied by the men engaged in taking the measurements.

With a view of determining the advantage, if any, of the double arrangement of fans, the belt of the right-hand fan was removed, and the top of its chimney closed as tightly as possible, and experiments were made on the left-hand fan alone, driven by the same engines, with the result of showing only about one-half the efficiency attained by the two fans running together, the single fan requiring 195 indicated horse-power to exhaust 275,000 cubic feet of air at 2-inch water-gauge, while the two together required but 155½ indicated horse-power for 337,000 cubic feet at 2½-inch water-gauge; the two showing a maximum efficiency of $85\frac{4}{10}$ per cent., and the single fan but $44\frac{5}{10}$ per cent. The single fan was evidently overloaded with air, as it was passing nearly 40 per cent. more than the cubical contents of its blades per revolution, and a very large amount of power must have been absorbed by its internal friction. Moreover, the shutter was regulated for the double fans, and its opening was too small for the single fan.

The very high efficiency recorded for the double fan led to the expression of considerable doubt as to the accuracy of these experiments, and in August the fan was again tested (Experiment C in the table), running at the same speed (100 revolutions), at which it had before shown its maximum efficiency. In the time that had elapsed there had been considerable changes in the mine, and neither the water-gauge nor the quantity of air was found the same as in April. This time the air-measurements were made in the mine at the foot of the 13 x 18 feet upcast-shaft. The double fan was found to be giving 298,000 cubic feet of air with a water-gauge of $2\frac{6}{10}$ inches, while the engines indicated $143\frac{42}{100}$ horse-power (averaged as before on twenty cards); the fans again showing, under changed conditions, an efficiency of $85\frac{2}{100}$ per cent.,—a very satisfactory check on the first observations.

The first four experiments (A, B, D, E) show a good experimental verification of the law that the water-guage varies as the square of the tangential velocity, the values of

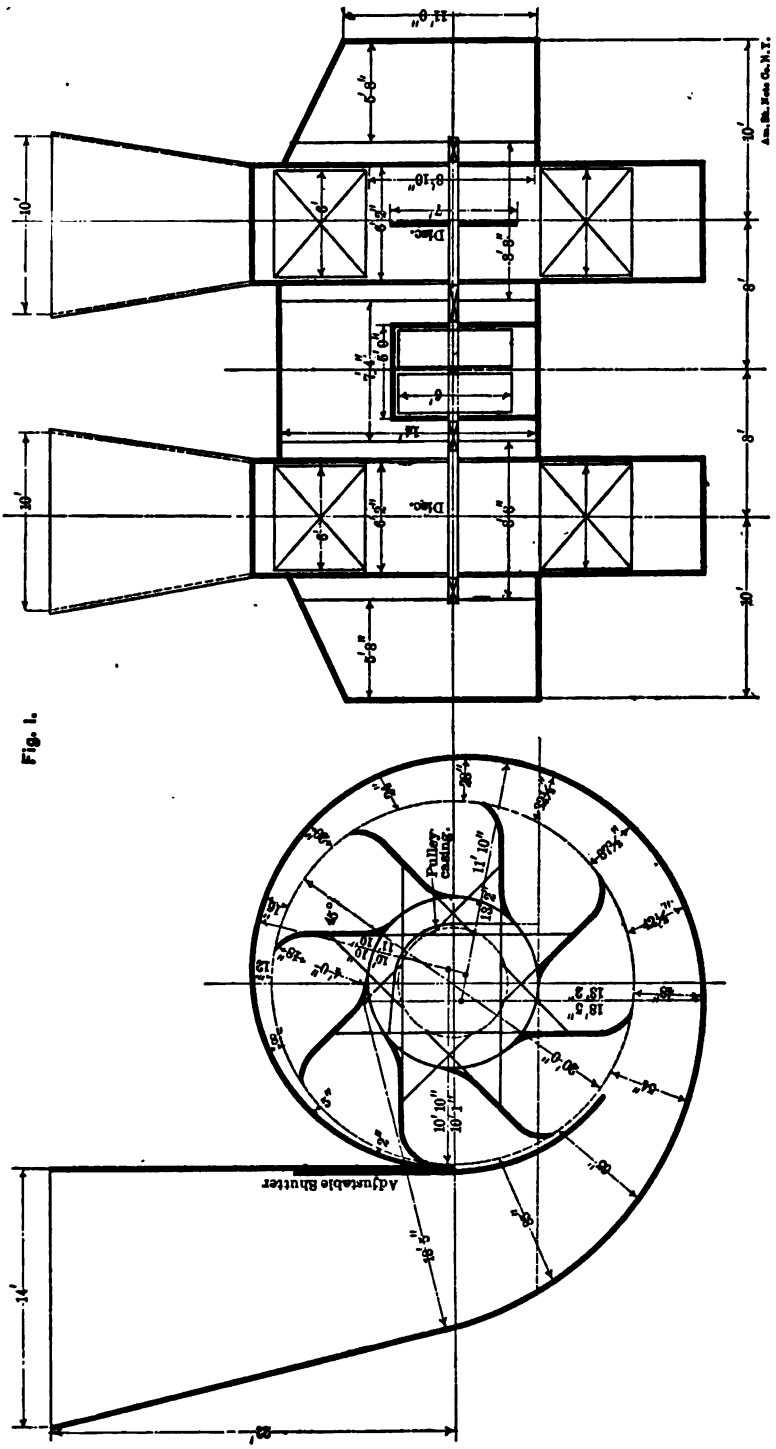


Fig. 1.

Double Fan of the Susquehanna Coal Company, at Nanticoke, Pa.

$\frac{WG}{T^2}$ being for experiment A .000000059

B .000000063
D .000000069
E .000000060

Showing a variation of only one hundredth-millionth part.

Fan No. 2 is a 20-foot single fan, ventilating the Susquehanna Coal Company's No. 6 slope workings at Glen Lyon, Pa. It is similar in construction to one of the double fans shown in Fig. 1, but without shutter or central disk. The curved sheet-iron on the inside of the blades is also omitted, and these have their tips curved forward instead of backward, their shape being as shown in Fig. 2. The fan is driven by two 16 x 24-inch engines, connected direct to either end of the fan-shaft.

This fan is doing the poorest work of all the Susquehanna Coal Company's fans hitherto tested. The air-ways in the mine are small, and very little air is needed; but the fan shows the unusual quality of attaining a higher efficiency at 3700 feet periphery-speed than at 5200. This anomaly is doubtless due, in a measure, to the engines, the valves of which were out of place at the time of the test, both giving very poor cards, especially at the higher speeds. In one engine, the back pressure was nearly equal to the mean effective pressure. The low efficiency may also be due to the absence of shutter and disk, or, possibly, to some trouble in the mine.

Fan No. 3 (Fig. 2) is a 25-foot fan, similar in general construction to Fig. 2, except that it has but one inlet. The chimney, also, is differently arranged, coming out at an angle of about 40° from the bottom, and having a shutter working through the upper part of the inclined chimney-structure. This fan, which ventilates the Susquehanna Coal Company's No. 2 shaft, Ross Seam workings, at Nanticoke, Pa., was tested at two speeds, 40 and 70 revolutions per minute. At the first of these, the fan was evidently moving too slowly for efficient work, though the $49\frac{2}{10}$ per cent. efficiency reported for fan and engine was rather due to the engine than to the fan, since a 24 x 32-inch engine, running at 40 revolutions and indicating only 13.82 horse-power, must consume a considerable portion of that in its own friction. That this was the trouble is further proved by the fact that at 70 revolutions, when the engine was developing $67\frac{1}{2}$ horse-power, the efficiency of fan and engine increased to over 82 per cent., making the fan, at this speed, nearly as efficient

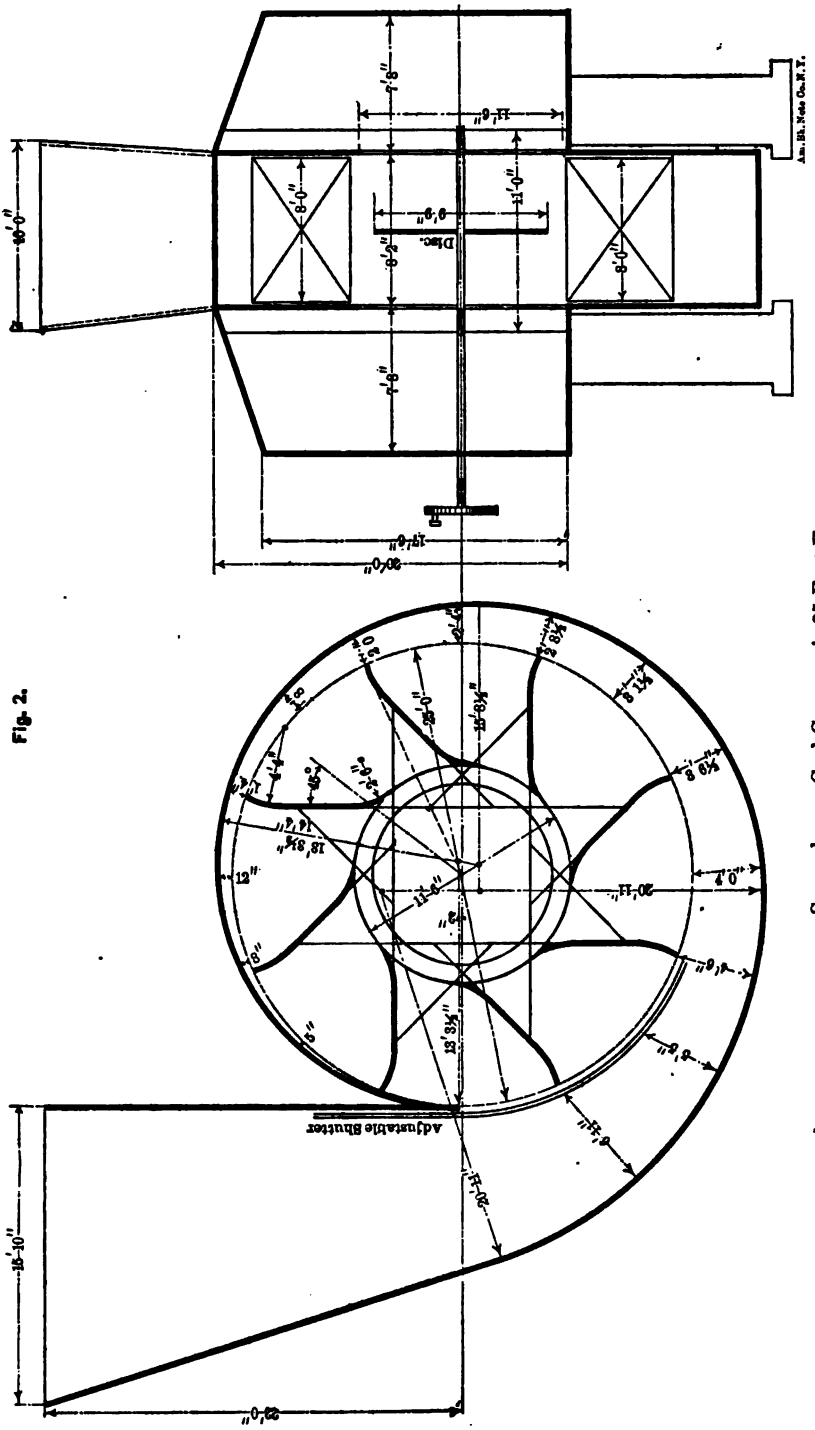


Fig. 2.

Susquehanna Coal Company's 25-Foot Fan.

as the double fan, No. 1, at its maximum, and more efficient than that fan at the same periphery-speed.

The seam which this fan ventilates is thin, and the workings are very much "gobbed up," and the fan is notable as the only American fan reported, ventilating a thin seam with an efficiency over 70 per cent. The air-measurements for this fan were made in the mine, near the foot of the air-shaft.

Fan No. 4, a 25-foot single fan, ventilating the Lee Seam workings, No. 2 shaft, Susquehanna Coal Company, at Nanticoke, Pa., is one of the oldest 25-foot fans in the possession of that company. Its construction is similar to that shown in Fig. 2, but it has only one side-opening, and the blades are flat, forming an angle of 45° with the radius. The fan was so old, that it was not considered safe to speed it beyond its regular rate. The air-ways in this mine are very large and clear, as evinced by its equivalent orifice, 75 square feet, and by the large quantity of air and low water-gauge. The efficiency of this fan, nearly 82 per cent., at so low a periphery-speed as 4867 feet, seems to indicate that, in this case at least, flat blades are no great disadvantage.

The air-measurements were taken in the mine, and, as in all the Susquehanna experiments, the water-gauge was taken at the fan-house door, and the power of the engine from the average of a number of Tabor indicator-cards.

Fan No. 5 (Fig. 3) is a double 17½-foot fan, ventilating the Baltimore tunnel workings of the Delaware and Hudson Canal Company. The two fans, each 4 feet broad, are driven on the same shaft by a single belt from a pair of 16 x 30-inch engines. The construction is remarkable for the shape of the *évasée* chimney, and for the very small breadth of the fans. The fans have neither shutter nor disc, and the blades are flat, bent back 45° from the radius.

This fan was tested in 1881; and the figures given in the table are from the mine-inspector's report for that year. No indicator was obtainable, and the horse-power was approximated by changing the pressure in the boilers, running with the throttle-valve full open, and assuming 75 per cent. of the gauge-pressure as the mean effective pressure. This, judging by cards taken from a number of similar engines, was very nearly correct, probably better for the higher than for the lower powers.

Fan No. 6 (Fig. 4) having 18 feet diameter and 6 feet 4 inches width appears to be enshrouded in a true but small spiral, and to have only an apology for an *evaseé* chimney, and it is remark-

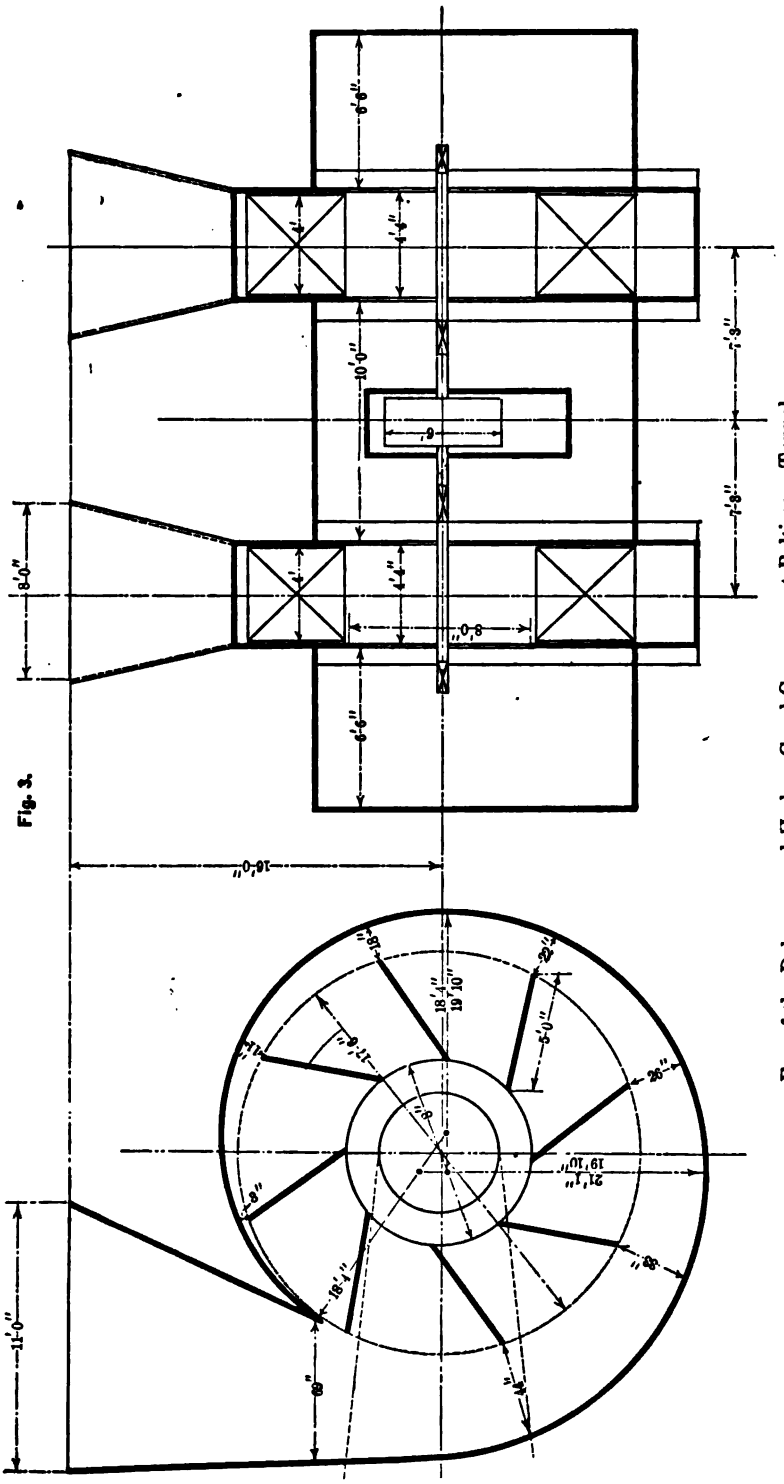


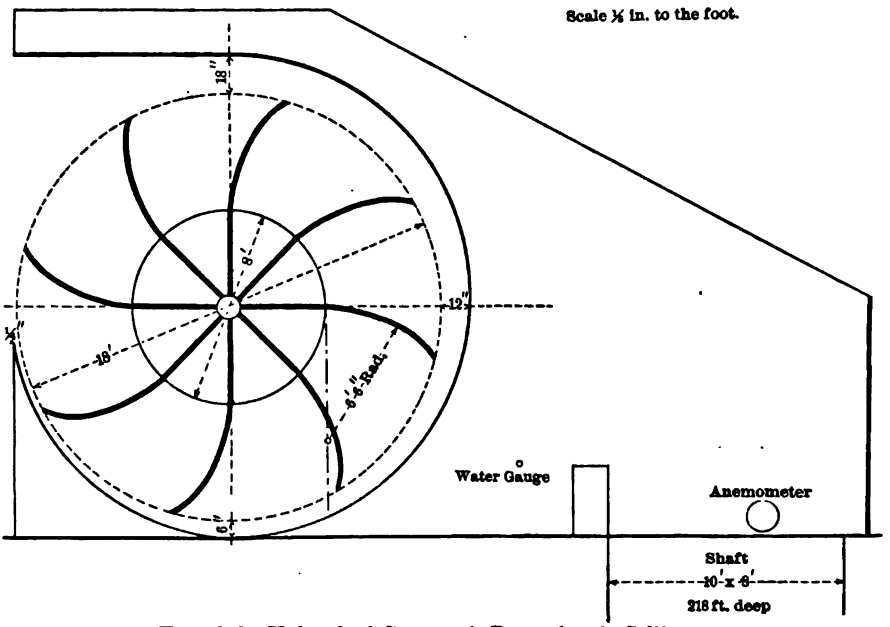
Fig. 3.

Fan of the Delaware and Hudson Canal Company, at Baltimore Tunnel.

able that it should give as good a result as seems to have been the case.

Fan No. 7 (Fig. 5) 18 feet in diameter and 5 feet wide, ventilating part of the Cameron Colliery of the Mineral railroad and Mining Company, Shamokin, Pa,, is similar in construction to No. 6, but with a good chimney. It passes so small a quantity of air that we may infer that the opening into the chimney is probably much too large, causing re-entries of air and lowering the efficiency of the machine.

FIG. 4.

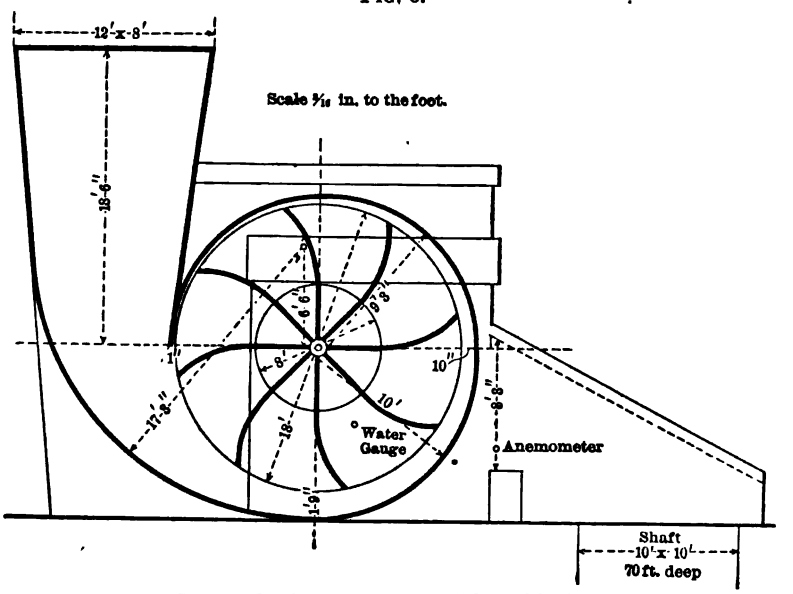


Fan of the Union Coal Company's Pennsylvania Colliery.

Fans No. 8 (Fig. 6) and No. 9 (Fig. 7) are in use for ventilating the Lehigh and Wilkes-Barre Coal Company's Stanton Colliery. They are arranged together, one working and the other held in reserve, to draw from the same air-shaft (11 feet 10 inches by 10 feet 6 inches in section), the connection to either fan being made by two large sliding doors either one of which can be raised, while the other is lowered by the same mechanism. No. 8, the older fan of the two, is about 20 feet only from the shaft to which its inlet is directly opposite. No. 9 is placed in front of No. 8 and connected to the air-shaft by a curved brick passage, with iron roof. Each fan is driven

by a 20 x 48-inch direct-acting engine, both engines being in the same engine-house. The air was measured by an anemometer attached to a long pole, moved about in the current (as described for fan No. 1), one four-minute measurement being checked by a two-minute one, and the two being found to agree very well. The air was measured directly in the opening of the fans, each of which had a clear opening of 230 square feet. This point was chosen on account of large leakages of the air-passages between the shaft and fan, estimated by the Lehigh and Wilkes-Barre Coal Company's engineer to be about

FIG. 5.



Fan of the Cameron Colliery, Shamokin, Pa.

8 per cent. of the whole volume delivered. These leakages may account for the discrepancy between the two fans both running at 46 revolutions per minute, and one giving 229,000 feet of air under 1.95 inch water-gauge, the other 235,000 cubic feet under 1.85 inch water-gauge. The fan-door, the only available place to take the water-gauge, was in the latter case in the current and the water in the glass vibrated considerably, so that it was difficult to get an accurate reading.

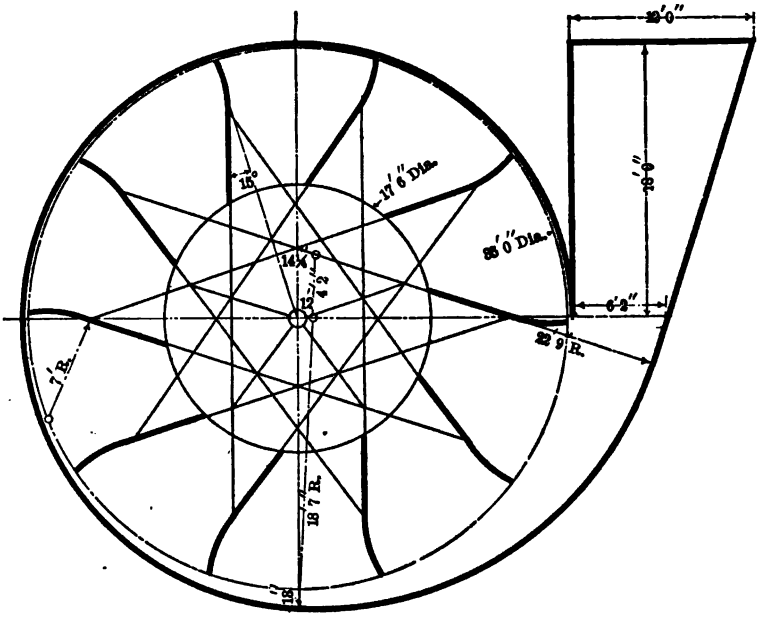
The lower efficiency noted for fan No. 9 at 55 revolutions may be partially accounted for by considerable strain noticed in engine and

fan at that speed, and partly by the very small air-shaft through which the fan had to draw.

Fan No. 10 (Fig. 8) ventilates part of the Empire mine workings of the same company; the air in this case was measured by the writer in the upcast-shaft (four 2-minute measurements agreeing within about 4 per cent.). The fan has no shutter, and is driven direct by a 18 x 36-inch engine, with a governor which prevented it from being speeded to over 73 revolutions.

Fan No. 11 (Fig. 9) is a Capell fan ventilating the South Wilkes-Barre shaft-workings of the same company. This fan is radically

FIG. 6.

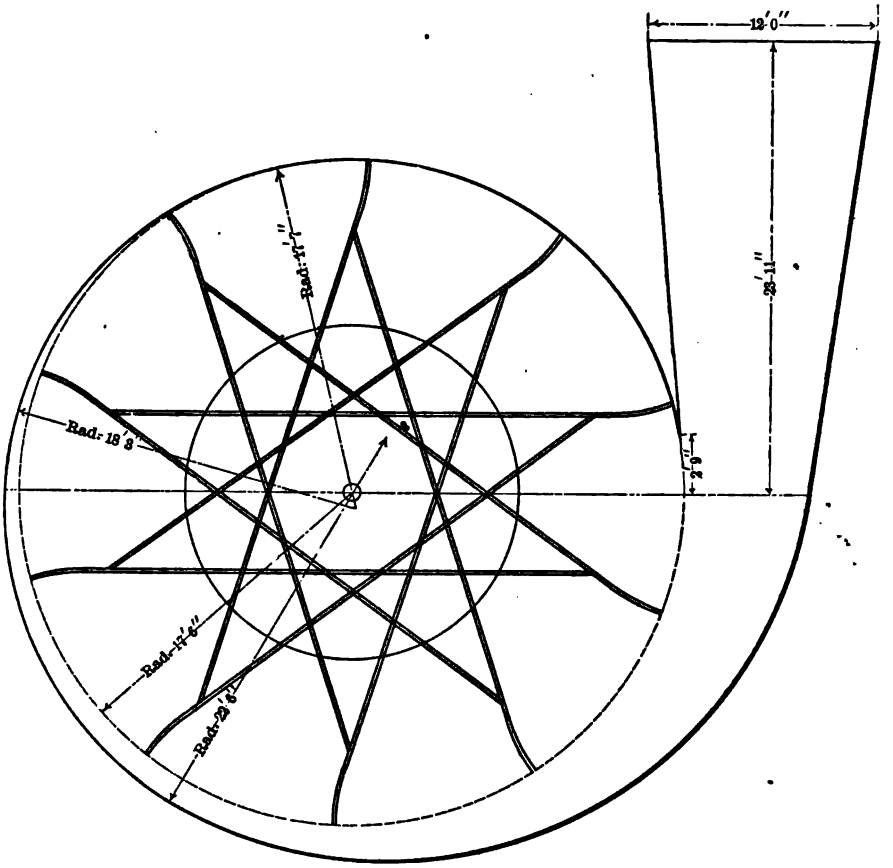


Fan of Lehigh and Wilkes-Barre Company's Stanton Colliery.

different from any other in use in the region. It is built entirely of metal, has six outside blades curving back nearly to a tangent, and six inside blades of about half the width of the outside ones and reaching to within about a foot of the shaft. The fan is 12 feet 6 inches in diameter and 12 feet wide, with two side openings. The sides beyond the inlets are closed, and the fan revolves in a large spiral casing. Fan No. 12, an English fan of about the same size, is inserted for comparison, and appears to give a very much higher efficiency. The officials of the L. & W. B. C. Co. state that their

fan is not having a fair chance to show what it can do, and from the record of No. 12 it seems probable that this form is better suited to mines with small air-passages, giving a much higher water-gauge. It is now working under conditions most favorable to the Guibal fan. When the workings at this place have become more extensive this fan should show much better results.

FIG. 7.

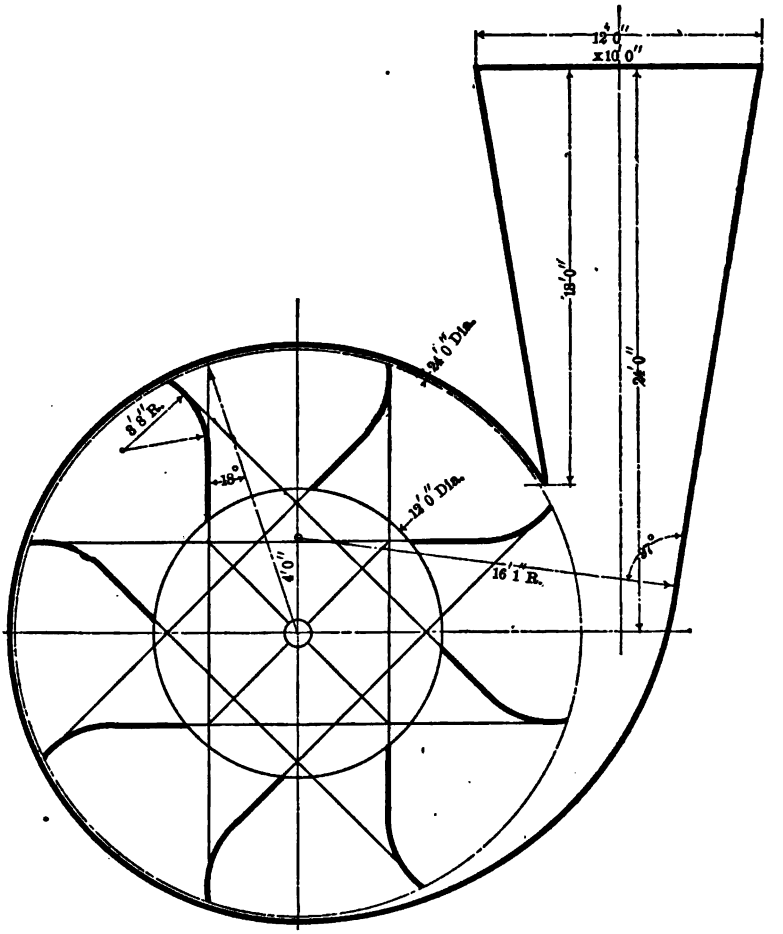


Fan of Lehigh and Wilkes-Barre Company's Stanton Colliery.

Fan No. 11½ (see Fig. 7) is a 35-foot Guibal fan ventilating the Hollenback mine-workings of the Lehigh and Wilkes-Barre Coal Company. It is reported as giving, at 45 revolutions, the enormous efficiency of $91\frac{3}{10}$ per cent.—much the highest of all the fans tested. The air was measured in the inlet of the fan, and the velocity ob-

tained from the average of three 4-minute measurements made by Mr. R. M. Snyder, engineer of the company. This is one of three fans ventilating the same mine, one of 18 feet and one of 35 feet diameter being in use in addition to the one tested. The air-shaft from which this fan draws is divided from the one used by the 18-

FIG. 8.

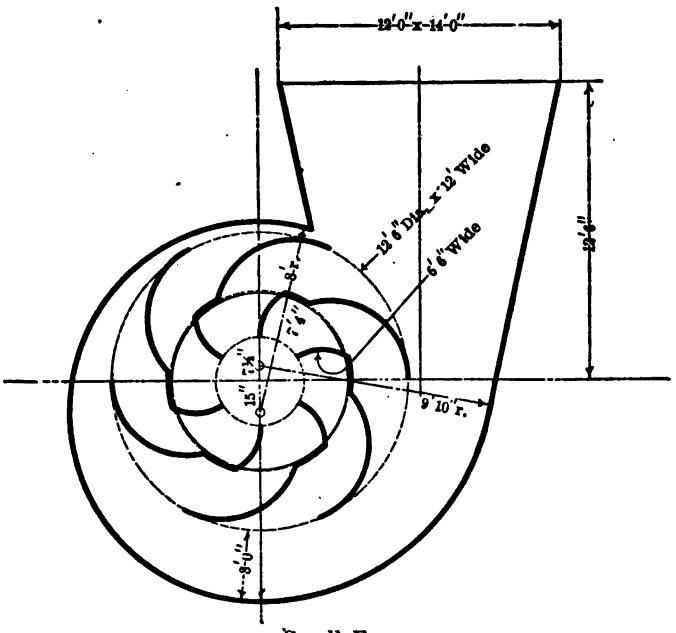


Fan of Lehigh and Wilkes-Barre Company's Empire Colliery.

foot fan by a wooden brattice only, and the air-ways are identical in the mine. The water-gauge was taken in the fan-casing on the side opposite the inlet, about three feet from the fan-shaft. The fan was tested at two speeds, 45 and 50 revolutions—the former showing $91\frac{3}{10}$

per cent. and the latter but $74\frac{2}{10}$ per cent. efficiency. The equivalent orifice of the mine, calculated from the two tests, is in the one case 58 square feet and in the other $65\frac{5}{10}$ square feet, and the water-gauge does not agree well with the square of the periphery speed. The heights at 45 and 50 revolutions respectively should have been 2.25 and 2.65, or 1.94 and 2.40 inches, instead of 2.25 and 2.40. This fan shows the same peculiarity as No. 9 in giving a lower efficiency at its higher speed. The variations in water-gauge and equivalent orifice of the mine may be accounted for by the peculiar condition of the ventilation—three fans drawing on the same mine, and this one alone being speeded up.

FIG. 9.



Capell Fan.

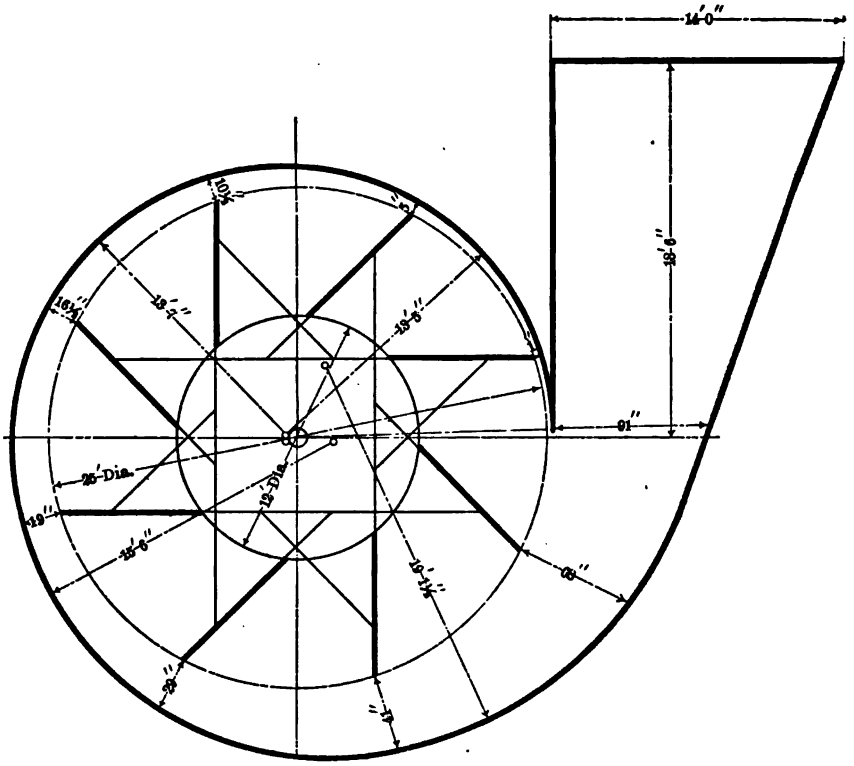
Fan No. 12 is an English fan, of which the record is introduced for comparison. (See under Fan No. 11.)

Fan No. 13 (Fig. 10) has been most exhaustively tested by Mr. Robert Ramsay, engineer of the South-west Coal and Coke Company at the Morewood mine, and the results are communicated by the makers.

Fan No. 14 (Fig. 7), located at the Dorrance colliery of the Lehigh Valley Coal Company, is a 35-foot fan exactly similar to

No. 9, except that the former is reported as 11 feet 8 inches and this as 12 feet wide. It is driven direct by a 24 by 42-inch engine. The reported efficiency, $76\frac{2}{10}$ per cent., is but a little higher than that of No. 9 ($72\frac{4}{10}$ per cent.), although the fans are working under different conditions, No. 14 giving 177,000 cubic feet of air under 2.25 inches water-gauge, and No. 9 235,000 under 1.85 water-gauge. The mine appears to have little influence on the efficiency in this

FIG. 10.



Fan of S. W. Coal and Coke Company's Morewood Mine.

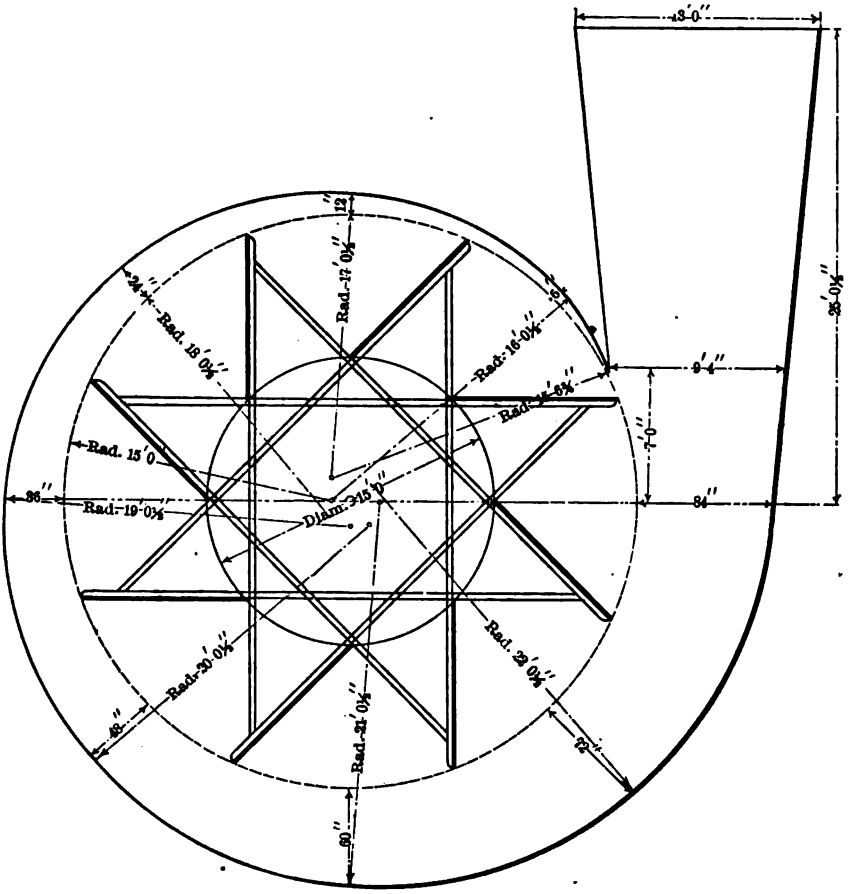
case. The air was measured underground at the foot of the up-cast, and the air-ways are stated to be as follows:

Split No. 1, 5000 feet long.	. . .	Regulator, 27 × 20 inches.
" No. 2, 800 "	. . .	" 16 × 12 "
" No. 3, 6350 "	. . .	" 36 × 36 "
" No. 4, 7950 "	. . .	Area, 120 square feet.

Fan No. 15 (Fig. 11), at the Lehigh Valley Coal Co.'s Henry

colliery, appears to give the high efficiency of $86\frac{55}{100}$ per cent. It is a 30-foot fan, 10 feet broad, with 8 flat blades inclined back 45° from the radius, a single intake, 15 feet in diameter, the stack starting inside 7 feet above the centre. The spiral casing begins at the point of cut-off and widens $\frac{1}{30}$ diameter for each 45° from the top,

FIG. 11.



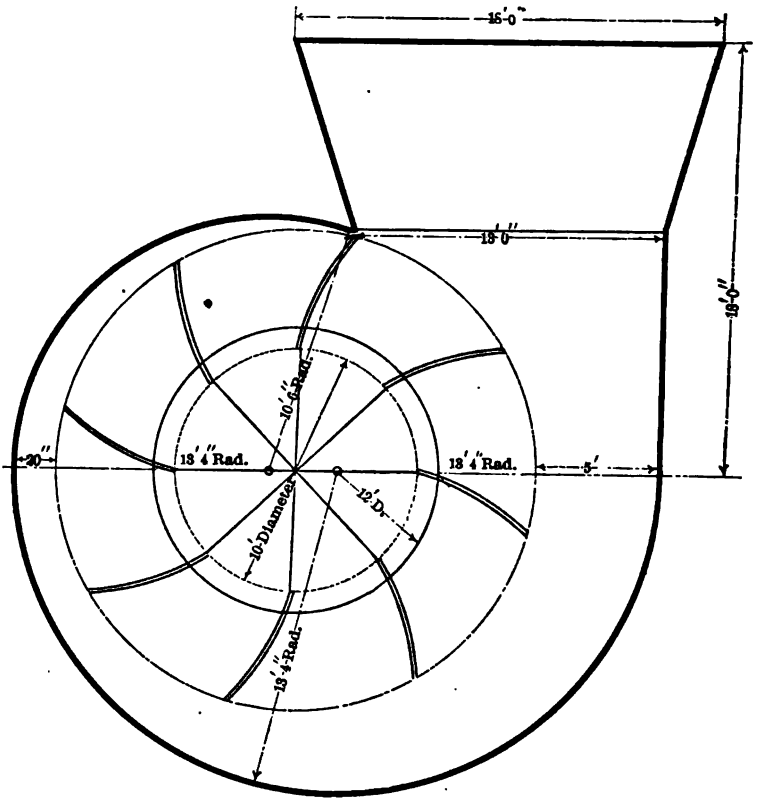
Fan at Lehigh Valley Coal Company's Henry Colliery.

beginning there at 12 inches and reaching 7 feet at the center-line. The air was measured in the mine. The airways of the mine are said to average 100 square feet in area, the current being divided into 7 splits, respectively 19,400 feet, 11,200 feet, 6600 feet, 11,400 feet, 13,200 feet, 20,200 feet and 2700 feet long. The equivalent orifice of this mine is about 84 square feet.

Fan No. 16 (Fig. 12), is ventilating the Lehigh Valley Coal Co.'s Packer No. 5 colliery. Its construction is, to say the least, unusual. The blades project 1 foot into the intake-opening, and both spiral casing and stack are odd in shape. The air-ways are said to approximate 100 square feet in area.

Fan No. 17, an English fan, is inserted in the table as having

FIG. 12.



Fan at Lehigh Valley Coal Company's Packer No. 5 Colliery.

probably the highest efficiency (over 80 per cent.) that has been reported publicly for any foreign fan. The Walker patent shutter (*Trans.*, xix., 37), is said to increase its ease of running rather than its efficiency.

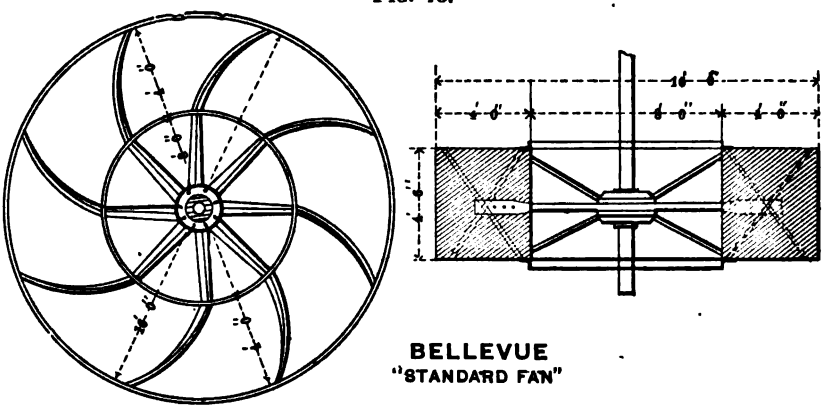
Fan No. 18 shows the efficiency, as determined 19 years ago, of one of the Delaware, Lackawanna and Western R.R. Co.'s open fans, employed at Avondale.

Fan No. 19 (Figs. 13 and 14), is the Bellevue fan of the Delaware,

Lackawanna and Western R.R. Co., which has been exhaustively tested by Mr. A. H. Storrs. The tests were made for comparison only, and the air was measured by an anemometer held stationary in the middle of the air-current. The anemometer was held always in the same place and gave results probably correct for comparison with each other, but not for absolute measurement.

A number of experiments on the difference between measuring air in the center of the current and averaging the whole (as described under fan No. 1), have shown that the former method gives results from 20 to 30 per cent. too high. Measuring in the center of each of four quarters of the space gives results from 8 to 15 per cent. too high. (See further remarks on page 662 in connection with Fig 15).

FIG. 13.



Original Form of D. L. & W. R.R. Co.'s Bellevue Fan.

This fan was originally (Fig. 13) open, 16 feet in diameter, and 4 feet 6 inches wide; and as such it was tested April 18, 1891, by changing the condition of the mine, giving various resistances to the air-current, with the results noted in the table (A, B, C). The fan was afterwards closed with a spiral casing and chimney (Fig. 14, spiral No. 1), and also provided for the experiment with a Walker shutter, the end of which is V-shaped, to reduce the vibration. The tests (January 29, 1891) of the fan thus arranged (experiment E) seem to show a gain in efficiency. The casing was then torn off and the open fan was again tested (February 2, 1891, experiment D), after which a casing and chimney of another shape (Fig. 14, spiral No. 2) were put on, and experiment F (February 12, 1891) was made. This seems to show the highest efficiency attained by the fan. The fan was arranged to

draw on a closed space while it was in each of the three conditions, and the water-gauge was taken (see G, H, I), giving for the open fan at 112 revolutions, 1.35 inches (G); for the fan closed, as in

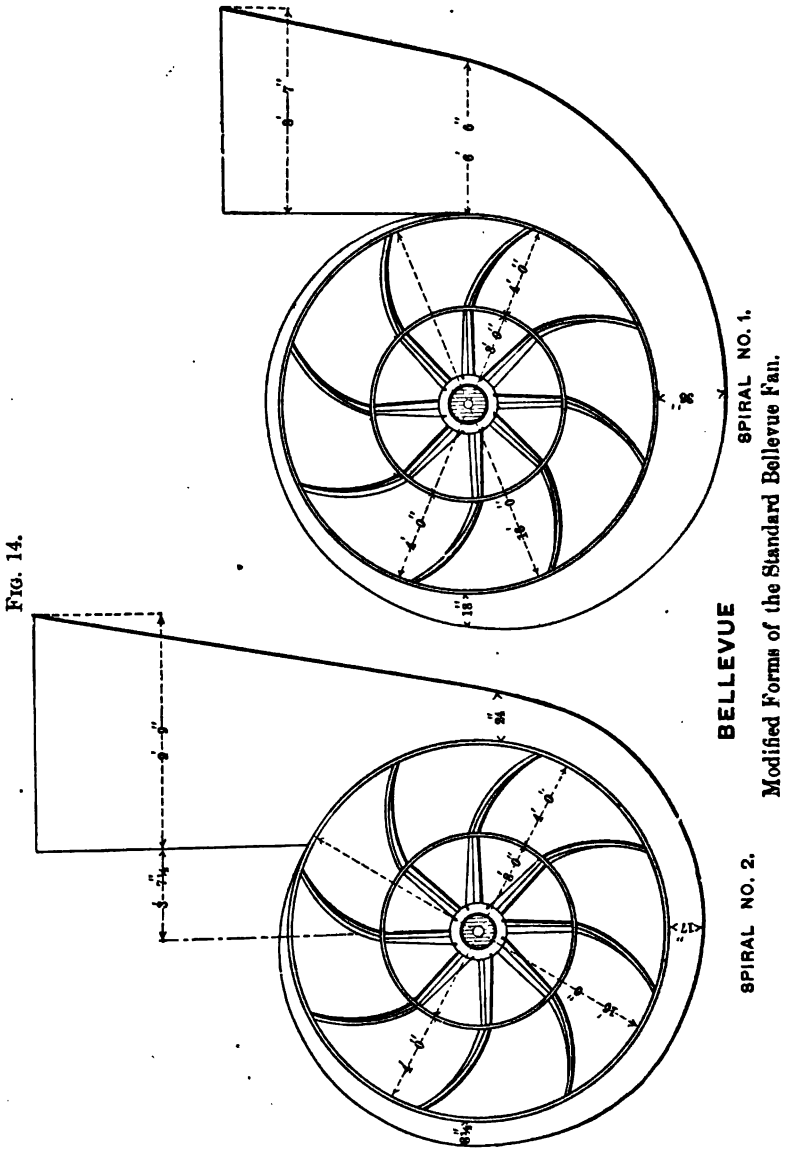
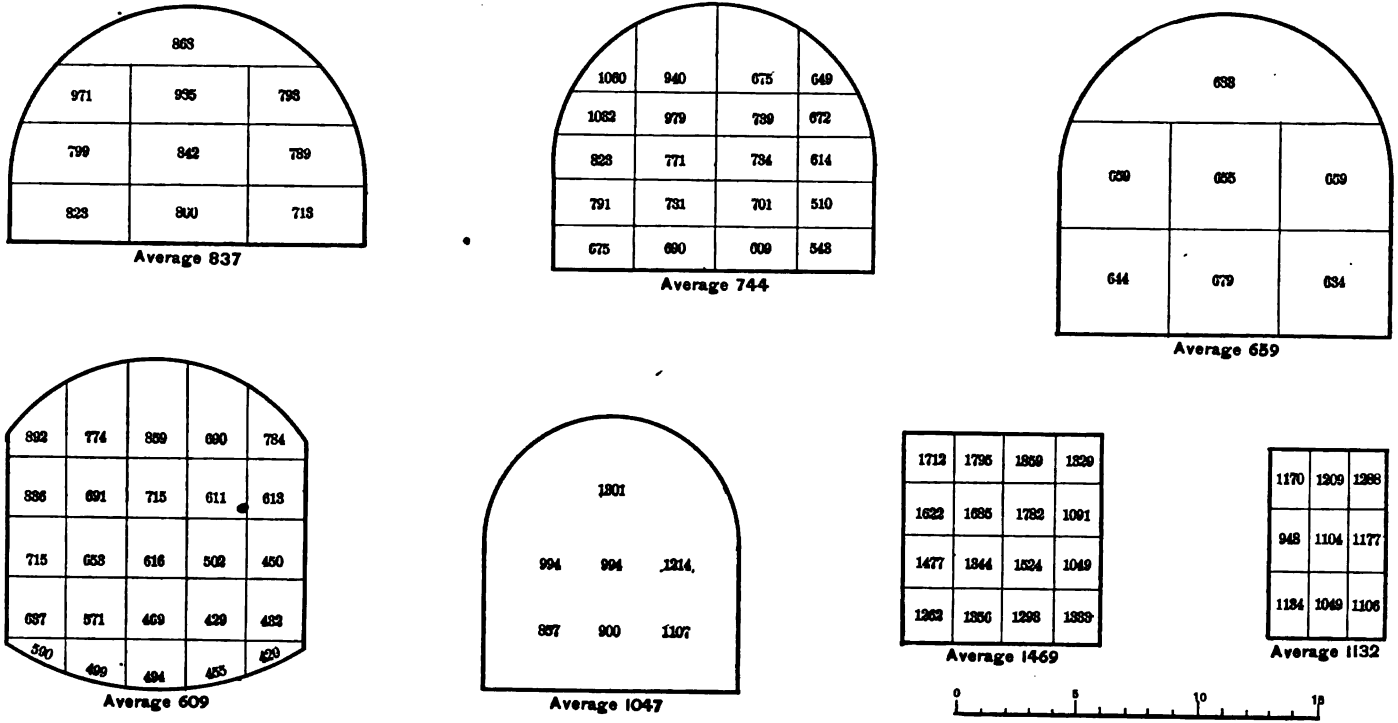


FIG. 14.

Fig. 14, spiral No. 1, 2.30 inches (H); and for the fan closed, as in Fig. 14, spiral No. 2, at $108\frac{7}{10}$ revolutions, 2.40 inches (I), or equivalent to 2.57 inches at 112 revolutions.

FIG. 15.

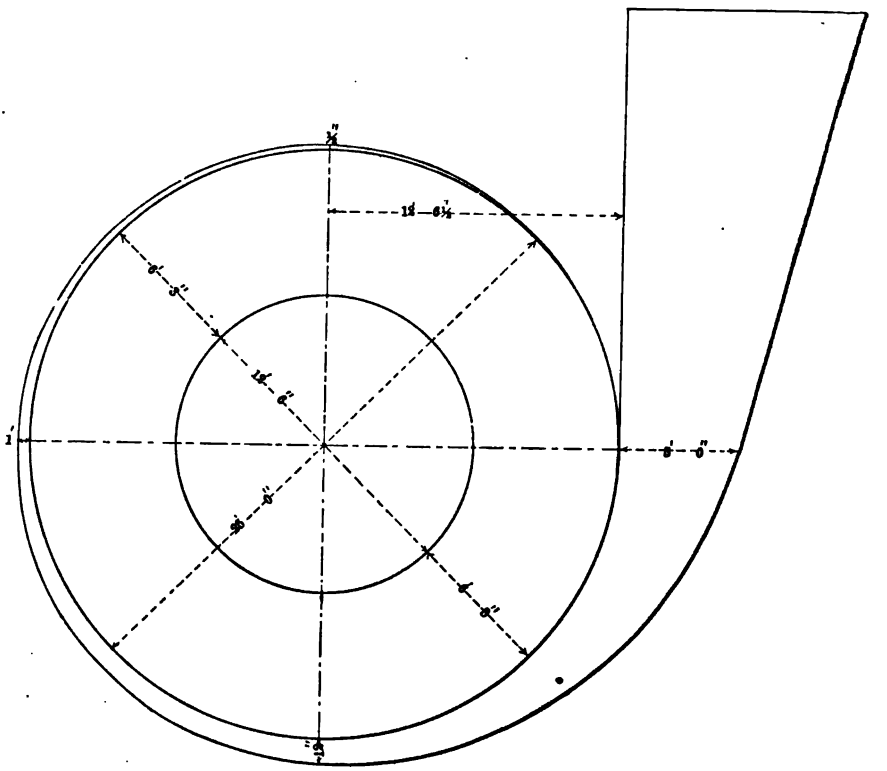


AIR MEASUREMENTS IN ENGLISH COLLIERIES.

Figures show velocities recorded in feet per minute.

Perhaps the three experiments, J, K, L, show the effect of the spiral casing, shutter and *évasée* chimney more reliably than the others. In all the three the fan was driven at the same speed (110.6 revolutions), showing for the open fan 136,000 cubic feet of air at 1.2 inches water-gauge; for the same fan closed (Fig. 14, spiral No. 1), 155,000 cubic feet at 1.6 inches, and with the addition of a Walker shutter,

FIG. 16.



Delaware, Lackawanna and Western R.R. Co.'s Holden Fan.

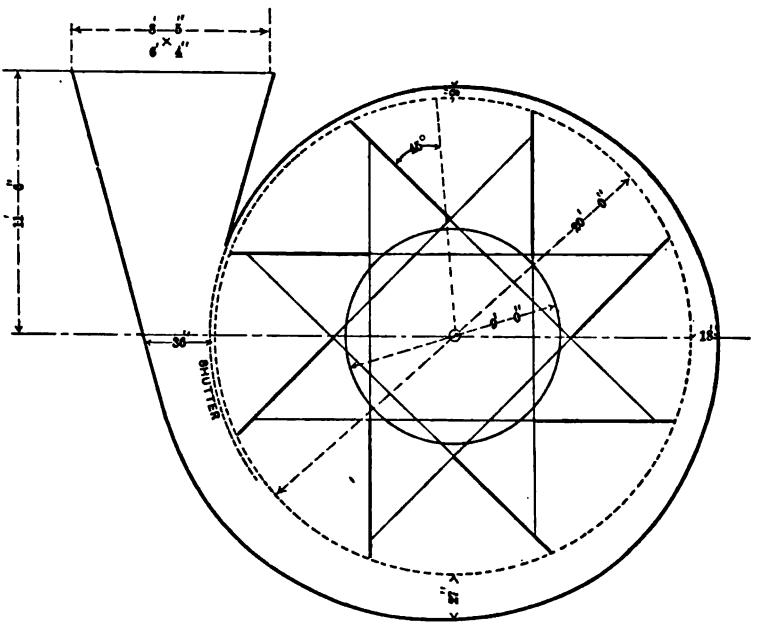
144,000 cubic feet at 1.5 inches. No indicator-cards being taken at this time, even the comparative efficiency could not be obtained.

The impossibility of even approximating the true quantity of air by measuring with an anemometer held stationary in but one position, is well demonstrated by Fig. 15, which shows the velocities of air found at different points in the cross-sections of seven different air-ways. These results are taken from a paper read by Mr. Wm. Daniel before the Institution of Mechanical Engineers, Manchester,

England, November 5, 1875. They show remarkable variations in velocity with different positions of the anemometer. The instruments used were one Biram and two Casella anemometers, and the velocities given have been corrected by formulas.

Fan No. 20 (Fig. 16), the Holden fan of the same company, was tested with the mine in three conditions: first, normal (A); then with cages put on the same level in the downcast-shaft (B); and lastly, with the air-doors open, short-circuiting the air, and giving the enormous equivalent orifice of 114 square feet (G).

FIG. 17.



Coxe Bros. & Co.'s Drifton No. 2 Slope Fan. Scale $\frac{1}{8}'' = 1'$.

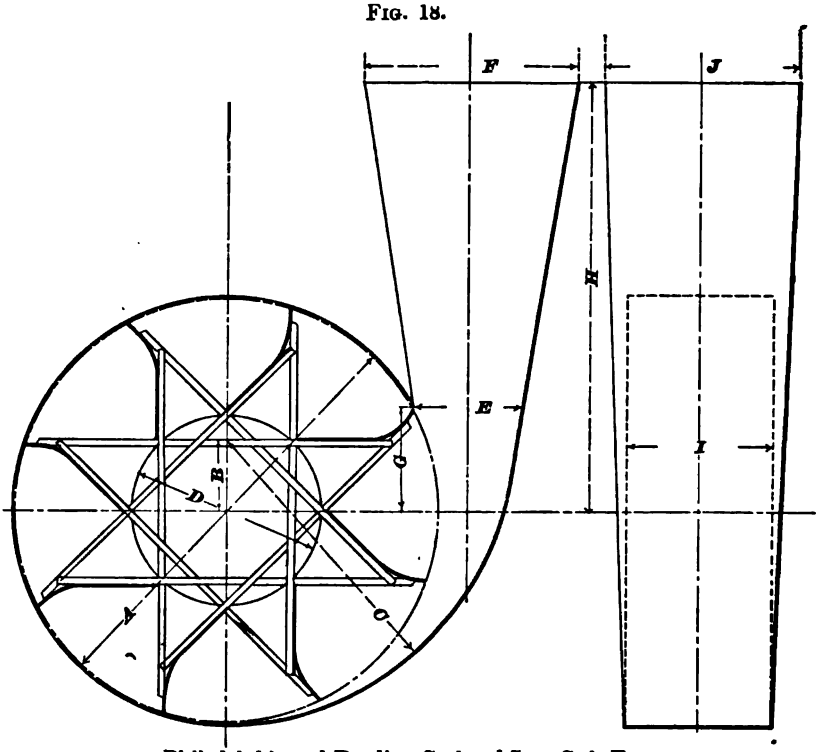
Fan No. 21 (Fig. 13), the Dodge fan of the D. L. & W. R.R. Co., was tested August 28, 1891, (A and B) with mine normal at different speeds of fan and (C) with air short-circuited. It gives a remarkable volume of air for so small a fan, emptying itself nearly three times in each revolution.

Fan No. 22 (Fig. 17), is located west of the No. 2 slope of Messrs. Coxe Bros. & Co., Drifton, Pa. It has a Walker shutter, and is driven direct by a 16 x 24-inch plain slide-valve engine. The air was measured "at highest section of up-cast, just in front of fan-inlet," and the water-gauge was taken at the center of the fan-inlet

by a $\frac{3}{4}$-inch pipe reaching to within 4 feet of the center. The size of the opening into the chimney appears to be small, and the fan is giving but little air; the fan most nearly comparing with it, No. 2, the poorest of the Susquehanna Coal Company's fans, giving more air under a higher water-gauge at three revolutions less per minute.

Fans Nos. 23, 24 and 25 (Fig. 18), of the Philadelphia and Reading Coal and Iron Co. are all made to a standard scale of propor-

FIG. 18.



Philadelphia and Reading Coal and Iron Co.'s Fan.

tion, based on the diameter of the fan. The dimensions are all proportional to the diameter.

Thus, A being the diameter of the fan:

$B = \frac{1}{6} A.$

$C = \frac{2}{3} A + 1$ inch.

$D = \frac{1}{2} A.$

$E = \frac{1}{4} A.$

$F = \frac{1}{2} A.$

$G = \frac{1}{4} A.$
$H = A.$
$I = \frac{1}{8} A + 2$ inches.
$J = \frac{45}{100} A.$

The width of the blades is $\frac{1}{3} A$.

Fans Nos. 24 and 25 are drawing from mines with very small orifices, each about 20 square feet; while No. 23 has a fairly large mine of 55 square feet orifice. The great difference in efficiency (24 per cent. for Nos. 23 and 24, and 60 per cent. for No. 25) is apparently inexplicable. The air was measured, as in the D., L. & W. R.R. Co.'s experiments, with anemometer held stationary, and consequently the reported efficiencies, etc., are not comparable with those of the other fans. The airways in the mine through which these three fans drew air are stated to be as follows:

No. 23.—Main fan air-way, 101 square feet, 4 splits, each averaging 6000 feet long, 30 square feet section; equivalent orifice of mine, 55 square feet.

No. 24.—Main outlet air-way, 64 square feet, 36 "monkey" airways or splits, averaging 2400 feet long each, 30 square feet; equivalent orifice of mine, $20\frac{6}{10}$ square feet.

No. 25.—Main fan air-way, 81 square feet area; 4 "monkey" air-ways or splits, average area, 30 square feet; average length of each, 300 feet.

An examination of the detailed results of each test in the accompanying table, shows a mass of contradictions from which it is exceedingly difficult to draw any satisfactory conclusions. Yet the following appear to be more or less warranted by some of the figures:

1. *Influence of the Condition of the Air-ways on the Fan.*—Mines with varying equivalent orifices give air per 100 feet periphery-motion of fan, within limits as follows, the quantity depending on the resistance of the mine:

Equivalent orifice.	Cu. ft. air per 100 ft. periphery-speed.	Average.
Under 20 square feet,	1100 to 1700	1300
20 to 30,	1300 to 1800	1600
30 to 40,	1500 to 2500	2100
40 to 50,	2300 to 3500	2700
50 to 60,	2700 to 4800	3500
60 to 70,	3300 to 5100	4000
70 to 80,	4000 to 4700	4400
80 to 90,	3000 to 5600	4800
90 to 100,		
Over 100 to 114,	5200 to 6200	5700

The influence of the mine on the efficiency of the fan does not seem to be very clear. Fans Nos. 1, 4, 5, 8, 9, 11½, 15, and 20, with equivalent orifices over 50 square feet, give efficiencies over 70 per cent.; while Nos. 3, 6, 10, and 12, with smaller equivalent mine-orifices, give about the same figures; No. 3 (efficiency 82 per cent.), having but 32 square feet equivalent orifice, while, on the contrary, fans Nos. 1 (F & G), 11, 13, 20, 21, and 23, with equivalent orifices of over 50 square feet, give lower efficiencies, as do Nos. 2, 7, 16, 18, 19, 20, 21, 22, 24, and 25, all drawing from mines with small equivalent orifices. Especially noticeable, in this connection, are Nos. 23 and 25; the former drawng from 50 square feet equivalent orifice with but 23 per cent. efficiency, and the latter from $19\frac{8}{10}$ square feet, with nearly 60 per cent. efficiency; and also Nos. 12 and 21, which increased nearly 10 per cent. in efficiency, when the orifice of the mine was enlarged by short-circuiting the air. It would seem that, on the whole, large air-ways tend to assist somewhat in attaining a large efficiency.

2. *Influence of the Diameter of the Fan on its Performance.*—This seems to be practically *nil*, the only advantage of large fans being in their greater width and the lower speed required of the engines.

3. *Influence of the Width of a Fan on its Performance.*—This appears to be small as regards the efficiency of the machine; but the wider fans are, as a rule, exhausting more air;—Nos. 1 (12 feet), 5 (8 feet), 8, 9, 11½, 15 (12 feet), and 20 (8 feet), all exhausting over 4500 cubic feet per 100 feet periphery motion. The two Capell fans, Nos. 11 and 12, both very wide, do not, however, approach these. The only wide fan which does not follow this rule, is No. 14 (12 feet wide), exhausting but 3353 cubic feet per 100 feet periphery motion. The Susquehanna Coal Company has recently erected a 20-foot fan, 12 feet wide, drawing air through two side-openings. The air-ways to this fan are in progress, and it is hoped that a thorough test of it can be made and reported in an appendix to this paper.

4. *Influence of Shape of Blades.*—This appears, within reasonable limits, to be practically *nil*. Thus, Nos. 3 (B), 8, 9, 10, 11½, and 14, with tips of blades curved forward; Nos. 4, 5, and 15, with flat blades, and No. 1, with blades curved back to a tangent with the circumference, all give very high efficiencies—over 70 per cent. The Lykens Valley Coal Company has two fans, identical in every particular except that the tips of the blades curve forward in the one and backward in the other; these fans can be made to draw air from the same air-way, and the figures resulting from such a test (which

have not yet been received) may help to elucidate the matter. The curving back of the tips appears to diminish considerably the vibration caused by the blades passing the point of cut-off.

An experiment bearing on this matter was tried on a small fan at the Empire Colliery, a number of years ago. The fan-blades are described as having about the shape of those shown in Fig. 13. They were removed and turned around, and then driven curved forward instead of backward. The fan gave, in both cases, the same amount of air, but no determination of the power used in each case was made.

Another fan was found to have a strong eddy in the inlet, the air, in places, apparently blowing out of the fan. On examination, it was noticed that the inside edges of the blades projected a couple of inches inside the circle of the inlet. These were cut off flush and the eddy ceased.

5. *Influence of the Shape of the Spiral Casing on the Performance of Fans.*—This appears to be considerable. The shapes of spiral casing in use fall into two classes, the first presenting a large spiral, beginning at or near the point of cut-off, and the second a circular casing reaching around three-quarters of the circumference of the fan, with a short spiral reaching to the *évasée* chimney.

Fans having the first form of casing appear to give in almost every case large efficiencies. Such fans are Nos. 1 (85 per cent.), 2 (68 per cent.), 3 (82 per cent.), 4 (82 per cent.), 5 (85 per cent.), and 15 (86 per cent.). Fans with casings between the two classes are Nos. 6 (70 per cent.), 7 (59 per cent.), 8 (71 per cent.), 9 (72 per cent.), 11½ (91 per cent., doubtful), and 14 (76 per cent.). To the second class belong Nos. 23 (23 per cent.), 24 (24 per cent.), 25 (60 per cent.), and 10 (75 per cent,). Nos. 13 and 22 have a spiral belonging to the first class, but very much contracted. They give only medium efficiencies. It seems probable that the proper shape of spiral casing would be one of such form that the air between each pair of blades could constantly and freely discharge into the space between the fan and casing, the whole being swept along to the *évasée* chimney. This would require a special beginning near the point of cut-off, enlarging by gradually increasing increments to allow for the slowing of the air caused by its friction against the casing, and reaching the chimney with an area such that the air could make its exit with its then existing speed—somewhat less than the periphery-speed of the fan.

6. *Influence of the Shutter on the Performance of Fans.*—This cer-

tainly appears to be an advantage, as by it the exit area can be regulated to suit the varying quantity of air given by the fan, and in this way re-entries can be prevented. It is not uncommon to find shutterless fans into the chimneys of which bits of paper may be dropped, which are drawn *into* the fan, make the circuit, and are again thrown out. This peculiarity has not been noticed with fans provided with shutters.

7. *Influence of the Speed at which a Fan is run on its Performance.* —It is noticeable that most of the fans giving high efficiency were running at a rather high periphery velocity;—Nos. 1 B (85 per cent.), 6282 feet; 3 B (82 per cent.), 5495 feet; 4 (82 per cent.), 4867 feet; 5 (85 per cent.), 5278 feet; 8 (72 per cent.), 5060 feet; 9 (72 per cent.), 5060 feet; 10 B (75 per cent.), 5504 feet; 14 (76 per cent.), 5278 feet; 17 (80 per cent.), 5273, seeming to show that the best speed is between 5000 and 6000 feet per minute. A notable exception to this is No. 15 (86 per cent.), 4241 feet. All these fans appear to reach a maximum efficiency at somewhere about the speed given and to decrease rapidly in efficiency when this maximum point is passed. This is notably the case with No. 1 (85 per cent. at 6282, and but 75 per cent. at 7727), No. 9 (72 per cent. at 5060, and but 58 per cent. at 6050), No. 11½ (91 per cent. at 4950, and but 74 per cent. at 5500), No. 17 (80 per cent. at 5273, and but 75 per cent. at 6780), and No. 13 reaching a maximum at about 5000 feet. On the other hand, No. 11, at speeds varying from 4800 to 5800, shows but 45 per cent., and No. 19, at the same speed, 5500 to 5700, is rather low in efficiency.

Among the curious results in the table may be noted the enormous drop in efficiency in experiments F and G on Fan No. 1, where one of the two similar fans only was running. In this case the fan had two side-openings, the one towards the centre being more or less clogged by the presence of the pulley-casing there; but neither this nor the presence of the shutter seems to suffice to explain a decrease of 50 per cent. in efficiency. Again, the behavior of Fans Nos. 9 and 11½, in each of which a small increase in velocity was accompanied by a large decrease in efficiency, and that of Fan No. 3, where an increased velocity gave a greatly increased efficiency, are puzzling.

It is probable that many of these variations are due to methods of testing. In further study of the subject it would be a great advantage to have standard methods devised of measuring air and water-gauge. The latter is taken by different observers at different

points, the fan-door being usually preferred, both on account of its convenience and because it is usually near the fan and pretty well shielded from the effects of the current. The water-gauge on Fan No. 11½ (91$\frac{3}{10}$ per cent. efficiency) was taken in the casing on the closed side of the fan opposite the single inlet, and it seems probable that it was affected by the current impinging upon it, and gave a false reading.

A satisfactory way of testing fans for comparison would be the following: Let the fan be completely stopped off from the mine, and a square wooden passage built—say 10 by 10 feet and at least 50 feet long, to give a steady, regular current—connecting to the open air. The opening into the fan-house should be 10 by 10 feet, and a recess large enough to shield from the current the men engaged in measuring air should be built out from the passage about 15 feet from the fan-house. In this passage wires should be stretched dividing it into not less than nine squares; and in measuring the air an anemometer-reading should be taken in each square, the anemometer being slowly moved, to average the speed.

The fan, drawing air from this passage, should be tested at different speeds, the power being determined by the indicator, or, better, by dynamometer, the air measured as above, and the water-gauge taken in the fan-house, as near the fan as possible, but out of the current.

The 10-by-10-foot entrance into the fan-house should then be reduced to, say, 75, 50 and 25 square feet, successively, the experiments being repeated for each opening, and, lastly, the entrance should be completely closed, and the fan tested for depression when passing no air. This method would give results entirely independent of the mine-conditions, and would permit all fans to be tested under conditions more nearly similar than would be the case were they put successively on the same air-shaft. Unfortunately, it is hardly practicable, except with new fans not yet in use. For perfectly satisfactory comparison its only drawback is the necessity of taking the power for engine and fan together. To meet this objection, if the fan is driven by belting, the engine should be run alone at the test-speeds, and its indicated power deducted from that developed when running the fan. For direct-acting engines this would, of course, be impossible; but the engine and fan-shaft could be run together before the fan-spiders were put on.

It is hoped that the experiments described in this paper may awaken interest in a somewhat neglected subject. In conclusion, I

desire to express my thanks to the many colliery officials whose skilful labors and friendly courtesy have made its preparation possible.

Discussion.

Eckley B. Coxe, Drifton, Pa.: We all owe Mr. Norris hearty thanks for bringing up this subject of fans. Our practice hitherto has been a sort of "go-as-you-please." We put up a fan, and if it does not satisfy us we put up a bigger one; then some one else who knows very little more about it builds an "improved" fan; and so on. No really systematic effort has been made to place the subject of fans upon a proper basis. The experience of my firm is not very extensive or precise. One of our fans, named in Mr. Norris's paper, is a pretty bad fan; we put a Walker shutter on it, which increased its efficiency, and did away with the vibration. Before that, one could hear the vibration two or three hundred yards away. We have never been troubled much with the question of ventilators, because we have worked so largely at the surface. With regard to the mechanical ventilation underground, coal-operators have been too much accustomed to think that steam costs them nothing, and the question has therefore been, not what was the efficiency of the fan for the power employed, but rather (until recently), how cheaply we could get something that the mine-inspector would allow us to work under because it gave the required quantity? I hope that this paper by Mr. Norris will lead us all to look more carefully into the subject. Mr. Wagner, our engineer, can state more particularly the results obtained from the Walker shutter.

John R. Wagner, Drifton, Pa.: We tried at three speeds the fan of which Mr. Coxe speaks, namely, at 50, 62 and 70 revolutions. The efficiency without the Walker shutter was 23 per cent. at 62 revolutions; and with a small opening we got less than we did without any shutter. With the shutter so modified that the opening was very much larger, we got 32 per cent., and with the openings still larger, 40 per cent. efficiency. At 70 revolutions, without shutter, we had reached the limit, owing to the vibration. It was not safe to run the engine at any greater speed. But after the Walker shutter had been put on, we were able to run up to 100 revolutions without any considerable tremor.

A. H. Storrs, Scranton, Pa.: We made at a D. L. and W. R.R. colliery a trial of a shutter of this type, in connection with Spiral No. 1 (see page 660, *aute*). Though this was not exactly the

Walker shutter, it followed closely the proportions drawn by Mr. Walker for larger fans. No test of the effect of this shutter on the efficiency of the fan was made. The quantity of air passed by the fan was apparently slightly decreased by the use of the shutter; but this could probably have been remedied by changing the casing-spiral so as to increase the chimney-opening. The noticeable effect of the shutter was the entire stoppage of vibration, which had become so great as to prevent the running of the fan at any higher speed. This cessation of vibration and consequent allowance of higher speed, which would probably be more noticeable on fans of greater diameter, is very important, and a strong argument for this type of shutter.

Mr. Norris has spoken of fans emptying themselves some two or three times per revolution. From the "cubic feet per revolution" and "cubical contents of fan-blades," as given in his table, we find that the enclosed fans empty themselves from *one-half* to *twice* per revolution, while the open fans are emptied from *one and three-quarter* to nearly *three* times,—this, for fans of both types, on mines covering the same range of equivalent orifices. One open fan, on a very large orifice, shows that it was emptied nearly four times, while a closed fan, on a still larger orifice, only shows *one and one-half times.* It will also be noted from the table that for the open fans the "cubic feet per 100 feet periphery motion" is greater, in proportion to the fan-width and equivalent orifice, than for the enclosed type. Notwithstanding this apparently free discharge of the open fans, they show very low efficiencies.

As illustrating the very large capacity of centrifugal fans to pass air, if the conditions of the mine are made favorable, a 16-foot diameter fan, similar to No. 19, at 130 revolutions, passed 360,000 cubic feet per minute, and another, of same diameter, but slightly wider and with larger intake circles, passed 500,000 cubic feet, the water-gauge in both instances being about one-half inch.

It would seem that the difference in equivalent orifice, showing a change in the condition of the mine, of the two tests of fan No. 11½, sufficiently explains the fact which Mr. Norris notes, that the water-gauges and volumes are not proportional to the speed. The condition of the mine being changed between the experiments, the volume and water-gauge must necessarily be affected thereby.

T. D. JONES, Hazleton, Pa.: The efficiency reported in some cases by Mr. Norris is larger than I have ever been able to determine by experiment. It seems to me that 91 per cent. is very large

indeed. My own experiments, recorded in the Pennsylvania Mine Inspectors' Reports from 1875 to 1881, did not show more than 60 to 65 per cent. I think the only perfect comparative test would be the trial of fans of different types, all upon the same shaft and under the same conditions. As Mr. Coxe has said, we want primarily quantity of air; and it is proper to make that the first requirement, the cheapness of fuel, and all other things, being taken into consideration. One way, in my opinion, to get both quantity and efficiency, is to give large area to the air-ways.

MR. NORRIS: The measurements of air for the Susquehanna Company's fans (of which the high efficiency has been questioned by Mr. Jones) were made at the same time that the indicator-cards were taken; and at that time the mine was entirely unoccupied, except by the men who were measuring. The Susquehanna fans were all measured by the same persons, and I believe this to have been the case also in the case of each of the other companies referred to in his remarks. As to the methods of measurement, I can only say that it was the same in those cases—at least those of the Susquehanna, the Lehigh and Wilkes-Barre and the Lehigh Valley companies. I have checked the measurements in several instances, getting results within 3 per cent.; but of course the figures given in my paper can not be regarded as other than approximate. The 91 per cent efficiency reported for fan No. 11½ is in my opinion very doubtful; the same fan driven only 10 per cent faster is stated to show but 74 per cent.

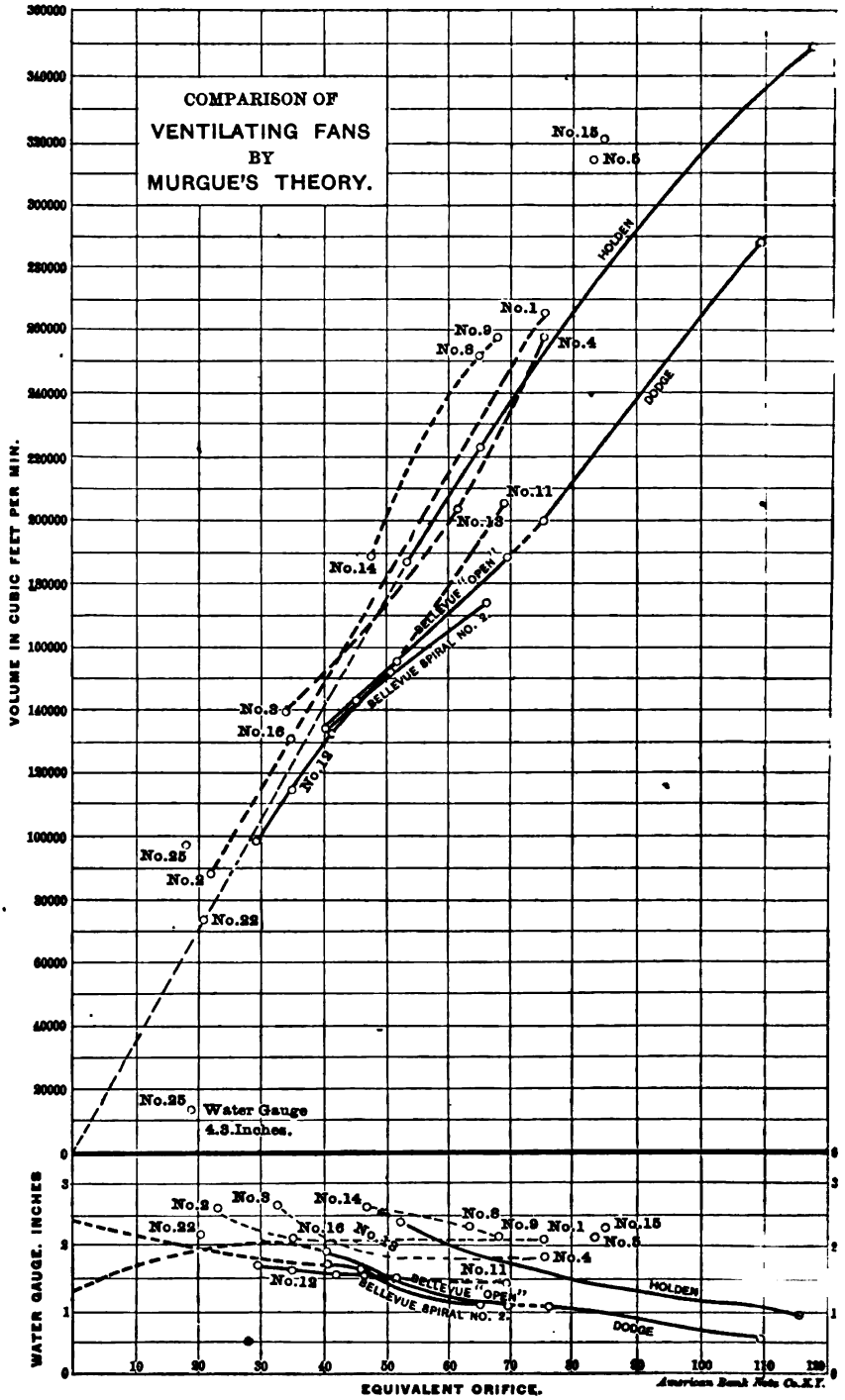
ROBERT ALLISON, Port Carbon, Pa.: The Institute is largely indebted to Mr. Norris for his paper; but it has not gone quite far enough. I take it that these ventilating fans are governed largely by local conditions, and that upon tests made in different localities with different fans, no comparison can safely be based with regard to their relative efficiency. I think the only way to get at this matter would be, as has been stated, to place a number of fans of different types on the same opening and test them all under the same conditions, with the same power applied, the same opening, and the same conditions all through. Then you could determine which would be the best form. Mr. Norris shows that where blades were turned around the efficiency was about the same. Now, a change in form of the casing might make quite a large difference. The form of the blades would make a difference. I know of a case where I built a fan and put the blades on with a curve backward, so that it would run back; but the superintendent of the mine did not like that form.

He had an idea that if the blades were turned the other way he would get better results, and so he took the blades out and turned them, and claimed that he got better results. If he was correct, then, of course, that is the way that fans ought to be built; but unless we make tests under the same conditions, it is impossible to arrive at specific results which will be valuable to the builders or users of fans.

I think it would be a good idea if a number of mine-owners and builders of fans would club together and raise a fund for the purpose of making a thorough test of the various forms of fans, under the direction of a competent committee of members of this Society. No doubt a suitable location could be secured for this purpose, and the result would have the greatest interest and value for mine-owners and builders of fans. Mr. Norris is entitled to our thanks for opening up this most interesting subject. The fan-builder, the mine-owner and operator, and the large number of men and boys employed in the mines are all interested in having the best possible ventilation for the security of life and property, and I hope this subject will not be allowed to rest until we have some good, reliable data at hand.

MR. STORRS: A fan of the type of Nos. 23, 24, and 25 in Mr. Norris's table, designed for a closed fan, was run for several years as an open fan, and run in both directions, with no perceptible effect on the amount of ventilation. At the request of the mine-inspector, the fan was enclosed, the enclosing spiral being copied as closely as possible after that of No. 25. The air was measured, both before and after the enclosing of the fan, while the colliery was in operation, so that great accuracy was not obtainable, but the measurements showed 3000 cubic feet per minute less after enclosure. This was a case where it appeared to make no difference in which direction the fan was run, or whether it was open or enclosed. No figures as to efficiency were obtained, but it is probable that the enclosed fan would have shown, in this respect, the best results.

With a view of comparing some fans by M. Murgue's theory of the equivalent orifice, with which Mr. Norris prefaces his paper, I have recently made some experiments with four fans of the D. L. and W. R.R. Co., putting the mines under different conditions so as to obtain data for each fan under as widely marked differences of condition of mine as possible. These observations of water-gauge and volume of air yielded were then reduced to the common periphery-speed of 5558 feet per minute, and the equivalent orifices


COMPARISON OF
VENTILATING FANS
BY
MURGUE'S THEORY.
VOLUME IN CUBIC FEET PER MIN.
WATER GAUGE. INCHES
EQUIVALENT ORIFICE.
No.15
No.5
No.1
No.9
No.8
No.4
No.11
No.13
No.14
No.3
No.16
No.12
No.2
No.25
No.22
HOLDEN
DODGE
BELLEVUE "OPEN"
BELLEVUE SPIRAL NO. 2.
No.25 Water Gauge 4.3 Inches.
American Bank Note Co. N.Y.

were calculated by M. Murgue's formula. With these equivalent orifices as abscissas, and the volumes of air and water-gauges as ordinates, two sets of curves were plotted, giving the characteristic curves of each fan.

From the data given in Mr. Norris's Table of Fan-Experiments, these same values were calculated for nearly all of the fans given in the table, and they have been plotted on the same diagram. Most of them being given in the table under but one condition of the mine, their curves could not be obtained; but it may be interesting to note that the various fans of about the same size and type form curves about such as might be expected for their type, the differing methods of air-measurements probably accounting for nearly all the irregularities.

The D. L. and W. fans tested were the Holden; a Guibal, 25 feet in diameter, 8 feet wide; the Bellevue "Open," 16 feet in diameter, 4 feet 6 inches wide; and the same fan with casing, called "Spiral No. 2," (see page 660), which also shows the casing and construction of the other two fans (Figs. 13 and 16); and the Dodge, exactly like the Bellevue "Open."

Referring to the diagram given herewith, it will be noticed that the curves of Bellevue "Open" and Dodge, if produced to meet, form one curve, with such variations only as might be expected from the method of air-measurement (see page 659). This gives a curve for the 16-foot open fan covering a wide range of conditions of the mine, represented by equivalent orifices varying from 43 to 108 square feet. A measurement taken on another fan of this type, giving an equivalent orifice of over 200 square feet, also falls on this curve produced, but outside of the limits of the diagram.

Although the efficiency of Bellevue Spiral No. 2 seems to be considerably higher than that of the same fan open, its curves would seem to indicate that an enclosed fan of its diameter was not large enough for mines giving as large equivalent orifices as Bellevue, for, as the orifice increases, the volume-curve is more widely separated from that of the open type. This Bellevue fan was tested, both Open and with Spiral No. 2, with the intake to the fan closed, giving the water-gauges shown with zero equivalent orifice and no volume. It will be seen that the enclosed fan gave a much higher initial water-gauge, and it will also be noted that the open fan, when working on an equivalent orifice of 43 square feet, showed higher water-gauge than when passing no air. Further tests of the open fan passing no air showed that the water-gauge did not increase in proportion to

the square of the speed, thus proving re-entrance of air between the fan-blades.

The Holden fan, with observations taken under mine-conditions varying from 54 to 114 square feet equivalent orifice, gives a very regular curve; but the volume for the large orifice could probably be increased by a change in the casing giving a larger discharge-chimney. The proportions of the casing of fan No. 15 are probably more nearly correct for such large volumes.

The high efficiency shown for fan No. 15, and the high position of its volume and water-gauge on the diagram, make it appear as the best of the fans given; and it is to be regretted that we have not observations of its work under other conditions of the mine.

Fan No. 12 is the only one in Mr. Norris's table which was tested under different conditions of mine, and with it we may also consider No. 11, which is of the same type (Capell) and about the same size. The water-gauge and volume of fan No. 11, plotted on the diagram, fall on the prolongation of the curve of No. 12. Judging from the diagram, and from the efficiencies given by Mr. Norris, this type of fan is not as good as the ordinary Guibal.

Taking up the remaining fans of Mr. Norris's table, and grouping those of the same size and similar construction, we get first the curves formed by Nos. 1, 2, and 16, all Guibals, 20 feet in diameter. Here, while there are some differences in the construction, notably the casing of No. 16, the curves which are obtained may be taken as those of this type of fan.

Nos. 3, 4, and 13 (25-foot Guibals), while they have some slight differences in form of blades, we should expect would form a curve closely approaching that of the last group; but No. 3 seems to be much better adapted for the work it is doing than the other two, and this makes the curve irregular.

Nos. 8, 9, and 14 (35-foot Guibals) have slight differences in the casings; and from the resulting curve it would seem that No. 8 was preferable to No. 9; but as the latter shows the higher efficiency, it is possible that there are errors in the air-measurements which would account for the variation in the curve.

No. 22, although of the same size as Nos. 1, 2, and 16, does not, on the smaller orifice, appear to do as well as we should expect of them. The shape of the blade is quite different, as is also the discharge-chimney, and probably both affect the performance of the fan.

Fan No. 25 appears to give remarkable results on very small orifice, and No. 5 also seems to be doing well.

From the diagram, there seems to be no doubt that the Guibal gives the best results of the three types taken up; but there appears such a wide variation among the performances of the various forms of this type, with very slight differences in blades and casing, that it will require a large number of experiments with these various forms to determine which is the best.

It seems, however, as though it must be perfectly clear that the method of comparison which Mr. Murgue has suggested is applicable to bring to a common basis the experiments at all mines; and that there is not the necessity, as seems to be the opinion of many of the prominent engineers of the coal-regions, of having all types and forms of fans tested on the same mine. If this method is accepted, there should be no trouble to obtain sufficient data from the different existing fans to warrant the determination of the best form.

It is hoped that the few comparisons here made may serve to show the value of this method, and that we may have experiments in the future not merely on various speeds of the fan, but with the condition of the mine changed, so that the results may show not only what each fan will do on its own particular mine, but on any mine. The comparison and discussion of these results should indicate the best form.

THE TESTS AND REQUIREMENTS OF STRUCTURAL WROUGHT-IRON AND STEEL.

BY ALFRED E. HUNT, PITTSBURGH, PA.

(Glen Summit Meeting, October, 1891.)

THIS paper is suggested by the appointment on the part of the American Societies of Mechanical and of Civil Engineers of committees upon the subjects of "Standard Tests and Methods of Testing," and "Uniform Methods of Tests of Material used in Metallic Structures," respectively, and by a study of the report of one of these committees, and other papers. Moreover, the writer has had frequent occasion to hear the complaints of manufacturers who furnish structural material subject to specifications, that in the tests prescribed, the number of tests required, and the methods of making

them and interpreting their results, engineers of design, who for the most part draw the specifications, do not sufficiently avail themselves of the experience of "the practical mill-men," and are not themselves sufficiently acquainted by experience with the details of the work to frame intelligently their requirements of the structural material that they order.

Most structural iron and steel is ordered from the mills by the bridge-maker, boiler-manufacturer, or other contractor, with whom the engineer only comes in actual business contact; and often between the engineer and the mill-manager another middle-man, "the commission agent," intervenes. The inspector under such contracts not infrequently hears the engineers who are the authors of specifications talked about as "tyrannical," "impracticable," or "insatiable" in their ever more and more rigid requirements, and as clothed with unjust power over the business rights of the manufacturers of structural material.

Such animadversions are injurious, and sometimes dangerous, to both interests involved. They are liable to lead to a lowering of standards of integrity among subordinates who have direct charge of the work of manufacture, tests and shipments of material. These subordinates sometimes gather from such incautious statements the notion that it is "smart" and for the interest of their employers and themselves to deceive the inspectors, and to ship material that will not answer the specifications, in the place of that which the manufacturer has honestly agreed to furnish. It need not be argued that such a course will bring upon iron and steel manufacturing concerns, no matter how large or powerful, disaster measured only by the failure of the structures to which their defective material may have contributed, and that decisive steps must be taken to eradicate both the fault and its cause.

On the other hand, by reason of the sharp competition and the difficulties of concerted action among manufacturers, there is often little scanning of the demands for tests and their requirements *until after the contracts have been closed;* and then the engineer, upon being appealed to for modifications, often looks upon such efforts very properly as attempts to lower the actual quality, and probably the cost, of the material to be furnished, matters which he thinks should have been called to his attention before the acceptance of the contract, but not after.

The iron- and steel-manufacturers and their mill-managements are more largely represented in the American Institute of Mining

Engineers than in either of the other societies referred to, who now have committees considering the subject of tests of structural materials, and it is hoped that such members of the Institute will contribute, either by discussion of this paper or by separate papers on the subject, the results of their experience. Their views will undoubtedly receive careful consideration, not only from the committees referred to, but from the engineering profession at large, which is watching with interest the progress of the work of those bodies. Very probably the subject of standard tests and methods of testing will come up for international discussion at the proposed Congress of Engineers in connection with the World's Columbian Exposition in 1893, which furnishes an additional reason for presenting at this time some suggestions concerning it.

The following propositions will be either immediately obvious or evident upon reflection:

I. The specifications for tests of structural material should furnish all necessary means for excluding all material deemed unsuitable by the engineer.

II. All testing costs money, which ought, either by special provision or by addition to the selling price, to be paid by the purchaser. In general, no doubt, this cost is practically added to (or, rather, included with other expenses in) the selling price by the manufacturer; consequently the number of tests and the severity of interpretation applied to their results should be limited by the engineer, just as far as this can be done without sacrificing the first principle above stated. This he should do for the sake of that reasonable economy which he is in duty bound to secure for his clients.

III. Besides the direct increase of cost by unnecessary tests and too rigid requirements there is another waste of money, through waste of time. Material rejected must be replaced before work can be completed, or perhaps even commenced, at the shops where it is to be fabricated into finished structures. Such delays are often tedious and expensive, even when the rejections causing them are entirely justifiable. Often the consumer must await the time when the proper rolls will again be in place for the same class of material; and rarely, if ever, do the terms of the contract secure to him adequate compensation for the, possibly very serious, losses thus occasioned through the vigilance of his own agent, the inspector. Of course, when excessive rejections and consequent delays are the result of unreasonable and perhaps ignorant or conceited severity on the part of the inspector, the mill-manager is tempted to regard with

great complacency the consequent embarrassments of an engineer or boiler-maker or bridge-builder under a time-forfeit.

The writer offers, from the standpoint of somewhat extended experience as an inspector of structural materials, some suggestions, directed to the following points:

I. What tests are usually made to determine the quality of structural material?

II. The number of tests required.

III. In what cases more than one test should be made or allowed before decision is reached.

IV. The way in which tests should be made.

V. The interpretation of the tests and test-requirements.

VI. What material the tests should cover.

VII. The jurisdiction of the inspector over rejected material.

VIII. The jurisdiction of the inspector over accepted material.

IX. The right and duty of the inspector to note and report upon methods of manufacture.

On all these subjects there has been much controversy, and there still remains fair room for discussion. The writer's views are submitted with the hope that divergent opinions also may be fully set forth.

I. What Tests Are Usually Made to Determine the Quality of Structural Material?

Tensile tests, including the elastic limit and ultimate strength per square inch, as measures of the strength and tenacity, together with the percentage of elongation and reduction of area as measures of the ductility, are agreed upon by all as the best physical tests, all things considered, to determine the quality of structural wrought-iron or steel. They are often necessarily expensive, owing to the cost of preparing test-specimens by means of planers, slotters, milling-machines, or lathe-tools, from sections which may be either too large or too irregular to be pulled in full size. In the writer's judgment, cheaper and more quickly prepared tests can be advantageously used as a substitute for *a part only* of these more expensive tensile tests. They have also the advantage of affording check and corroborations, not likely to be subject to errors that might continuously occur with one method of testing.

The determination of the modulus of elasticity in connection with tensile tests adds much to the time, labor and skill required, and is now seldom required as a measure of the quality of the metal. It is

generally used only as a special test in determining the applicability of varying materials which are to be used together in the same members of a structure.

The other tests, commonly used in the best practice to-day, are outlined below.

Annealed tensile tests of forged work, and of all material subject to subsequent annealing after test-specimens have been taken, are now called for. That is, a series of tensile-tests is required of specimens carefully annealed before being prepared for pulling in the testing-machines; these tensile results to be compared with those of tests made upon specimens in their original condition.

This seems to be an excellent provision, and the writer would suggest that, for tension-bars, of low steel at least, a suitable number of annealed test-specimens should be pulled in tension, to ascertain the normal tensile strength, since a considerably higher strength may be imparted to steel by cold-rolling, but will be lost in such a subsequent annealing as bridge-members receive after the eye-bars are made up. To determine the tensile strength which can be depended upon, therefore, the test-bars should be pulled annealed as well as unannealed.

The drifting-test, by striking with a sledge upon a steel drifting-pin in punched holes, and noting the size to which these holes can be enlarged, under different circumstances, without fracture of the material, is a practical test that shows the actual behavior of the metal, and can be made upon scrap pieces without wasting merchantable stock, and without expensive machine-shop work upon the test-specimens. A hole punched for a $\frac{3}{4}$-inch rivet, its center being $1\frac{1}{2}$ inches from the rolled or planed edge, is required to be capable of enlargement in this way without fracture of the metal, until it will pass a rod of the diameter of 1 inch for wrought-iron, $1\frac{1}{4}$ inches for bridge-steel and $1\frac{1}{2}$ inches for boiler-plate steel.

The writer believes this should be one of the standard tests of wrought-iron and steel plate and shapes. It presents the further advantage that exact data can be obtained for records regarding it, and, when desired, specimens themselves can be kept conveniently, stamped with the melt-number and other identifying marks, as a record of the quality of the material. This test is not upon the list of standard tests proposed upon pages 514 and 515 of the "Report of the Committee on Standard Tests and Method of Testing," of the American Society of Mechanical Engineers.

The bending-test for soft structural steel is, by itself, not a crucial

est of quality. I have found many specimens that, pulled in a testing-machine, were very brittle and crystalline in fracture, yet under the bending-test the bar would come over upon itself through an angle of 180° without fracture. Most steel, of less than 68,000 pounds tensile strength, and under ¾-inch thick, will bend down double over itself, if the specimens have planed edges, and sound specimens are taken to start with. At the same time, the bending-test is properly regarded by engineers as one of the important corroborative tests for both wrought-iron and steel.

The nicked bending-test for wrought-iron is believed to be important, and, in all good inspection, it is regularly made and carefully recorded as an evidence of quality. The conditions, however, under which nicked bending-tests are made, have much to do with the results. These conditions should be made as nearly as possible uniform, in order that their interpretation may be facilitated. The temperature at which the nicked bending fracture is made is an important factor. At low temperatures, as has been proved, most irons appear more crystalline than they do under ordinary summer temperatures. It need hardly be remarked here, that the nicked bending-test is not applicable to steel except under special conditions, as in rivets.

For rivet-rods, one of the best tests is actually to rivet up some work with them, then cut the rivets out, and see that they cut out tough and silky in the case of steel, or fibrous in the case of iron. Rivet-steel rods should not have over 60,000 pounds per square inch tensile strength, and should forge successfully at a red heat.

Where steel scrap is used with iron, in wrought-iron plates and shapes, the nicked bending-test, nicking to various depths of the section before bending, is usually the best way of ascertaining the soundness of the welds, and may be recommended for frequent employment.

The hardening- or quenching-tests of low-carbon structural steels prescribed by the Committee of the American Society of Mechanical Engineers (page 14 of the Report), seem to the writer to be of doubtful utility, except when very carefully made, and upon steels likely to be subjected to similar treatment in actual practice, such as fire-box steel, some steel rivets, etc. In my experience, some data of which are published in *Transactions*, xii., pp. 315, 316, all steel, no matter how low in carbon, will harden more or less, as shown by tension tests in a testing-machine; the amount depending upon not only the "hardness" in the steel, but also the temperature to which

it is heated and the character of the solution into which it is plunged to quench it. Unless all these conditions be very carefully kept uniform, the general experience is, that conflicting results will be obtained from quenching-tests.

In this connection, it may be pointed out that the test prescribed in some specifications that steel, having a tensile strength, in many cases, of from 64,000 to 70,000 pounds per square inch, and in some cases of from 60,000 to 68,000 pounds, shall be capable in test-specimens of bending double on itself after being heated to "an intense red heat," or to a "cherry-red heat," and then plunged into water, is very unsatisfactory, because the temperature to which the specimen has to be heated is difficult to determine, and yet small differences in this temperature have much to do with the endurance of the steel under the test.

The bulging-test recommended by Kirkaldy, though not often used in this country, is a useful practical measure of the ductility of the metal, and especially adaptable for testing of soft flange-steel. A sheet somewhat larger than the diameter of the dies being pressed through them, the energy actually expended and the amount of bulging of the plate through the dies under the given pressure, give an excellent measure of the metal's ductility. The pressure upon the dies being exerted by the moving plate, or plunger, of an ordinary testing-machine, its amount can be measured. This is a very easy, quick, and not at all expensive test, and the actual specimens may be preserved as a valuable record of the facts.

Compression-tests are not usually called for as methods of determining the quality of wrought-iron or steel. Since the factor of ductility does not largely or primarily enter into such tests as commonly performed, they are by no means crucial or conclusive as to quality. Metal like cast-iron, for instance, will give better results under compression than good soft steel.

Cold-forging tests have been included in some specifications which have required that a bar one-half inch square be found capable of being forged down cold to a thin flat edge. Much depends on the skill of the smith who does the work; and as there are easy ways of evading the requirements of the specification, the test is seldom used, and the writer believes it is of doubtful utility.

Hot-forging tests of metal to be used for special purposes, and as direct practical trials of the fitness of the metal for the work required of it, are often employed, and seldom objected to.

The welding-test is another practical trial of wrought-iron gener-

confessed to be reasonable where the metal is destined to be alded in actual use.

Kirkaldy, in his new work on testing, says that comparisons of *specific gravity* furnish an accurate method of determining the relative qualities of structural material. The writer's experience has not been very large in this matter, as the specific-gravity test is not an ordinary one for structural steel and iron; but from that experience he believes that in order to compare and generalize upon such determinations, the observer must have made them upon specimens treated under exactly similar conditions. This condition being fulfilled, specific-gravity determinations, being relatively inexpensive and quickly made with the proper apparatus, may be found excellent tests of structural material. The subject certainly deserves further consideration.

Specifications for most kinds of structural iron and steel which require, in addition to tensile and other physical tests, full *chemical analyses* of the metal, showing a given chemical constitution within very narrow limits of variation, are manifestly too severe for reasonable economy, except in special cases, where the importance of securing a metal exactly like previous standards overbalances any question of the considerably increased expense involved. Steel for the machinery of locomotives and other rolling-stock is an instance in point.

Inevitable variations in the raw material, the process of manufacture, and the amount and methods of manipulation from the pile, bloom, or ingot to the finished material, preclude the certainty of obtaining always the same tensile tests with the same chemical analysis. For example, a considerably higher percentage of carbon is required in the very low-phosphorus boiler-plate and other structural steel made by the basic process to-day, to give the same tensile strength as that of similar metal made by the acid process. The same remark applies to open-hearth steel largely made from soft direct-made bloom-iron, such as the product of the Carbon Iron Company of Pittsburgh, or from soft puddled blooms, as compared with the metal made from hard steel-scrap stock. Except where the larger deflection under transverse loads becomes a serious factor, the softer steel is better, and I believe it can be more safely trusted under higher unit-strains than the metal with a greater strength and greater apparent factor of safety. In this particular, the common engineering practice and reasoning seem to me erroneous, especially where the metal is to be heated and forged locally, as in eye-bars, or where it

is to be subject to alternate heating and perhaps sudden cooling, as in fire-box steel. In such cases, steel below 64,000 pounds per square inch in tensile strength can be safely trusted, in the writer's judgment, with a higher unit-strain than metal of higher ultimate strength obtained by running the carbon over 0.18 per cent.

As outlined above, there are reasons which may justify, under some circumstances, where high ultimate strength is required, the prescription of a carbon-test with a maximum allowable limit. It should be said, also, that very soft steel, as ordinarily made by the open-hearth or the Bessemer processes, becomes seamy, open, and unsound if below .07 per cent. in carbon. Again, in some cases, where the metal will be subjected to considerable forging, a sulphur-requirement, with a maximum of .04 per cent. for steel and .05 per cent. for wrought-iron, is advisable. But, under ordinary conditions, this matter, as well as that of the other chemical constituents, with the exception of the phosphorus in steels, can be safely and most wisely left to the manufacturer's own judgment of his product, and of the chemical formula that will (together with skilful manipulation in manufacture) give the required physical qualities. With regard to phosphorus, however, there is strong reason for specifying and limiting it in structural steel, for it has been found that more than 0.10 per cent. of this element greatly increases the chances of failure under the physical tests, while steel containing less than 0.08 per cent. of phosphorus gives much more uniform results physically than higher-phosphorus metal. Since the percentage of phosphorus in the stock from which acid steel is made determines largely (in inverse proportion) its relative cost, the manufacturer is tempted to let the phosphorus run near the danger-line, and often this economy is carried to a point that involves a positive loss, by reason of the increased amount of rejections thus occasioned. In order to put all parties upon an equal basis in bidding, as well as to secure a safeguard against cold-short metal, a maximum phosphorus-limit is now almost universally made for steel, 0.10 per cent. being the most lenient, 0.08 per cent. the more rigid requirement for acid steel. For basic steel, it has been considered wise to determine the phosphorus in each melt in order to know that the basic slag has performed its function properly. Many engineers demand less than 0.05 per cent. in basic steel where they allow .08 per cent. in acid steel of less than 68,000 pounds per square inch tensile strength. This extra requirement is based upon the supposition that, unless very high-phosphorus ferro-manganese be added, or there be some improper treatment in the last

of the heat, which restores a portion of the phosphorus to the al, phosphorus ought to fall below .05 per cent. in properly-made sic open-hearth or Bessemer steel of less than 0.20 per cent. carbon.

A considerable number of specifications made within the past few years for bridge-steel have discriminated against Bessemer steel, engineers preferring to pay the larger price for the greater average uniformity of open-hearth metal. Many specifications give a preference to open-hearth steel by fixing the phosphorus-limit at .08 per cent. for acid open-hearth and .05 per cent. for basic open-hearth and for Bessemer, whether acid or basic. This, as explained above, is not, however, a stipulation against basic open-hearth steel, for it has been found that no rejections are occasioned by the .05 per cent. limit for such metal.

A phosphorus-limit in wrought-iron has been sometimes called for; but it is only when the amount actually existing in the metal as a phosphide is separated from the phosphorus existing as a phosphate in the contained slag of the wrought-iron that any intelligent discrimination can be based upon the phosphorus-test. In the writer's opinion, it is not required as a means of distinguishing good structural wrought-iron from bad, and is not to be recommended. In most cases, where it is found in a specification, the author has probably reasoned by analogy that because the phosphorus-requirement is wise for steel, it should be applied to wrought-iron. This reasoning fails to recognize the chemical difficulty of accurate determinations, and ignores the practical sufficiency of the nicked bending-tests for indicating cold-shortness in wrought-iron.

II. The Number of Tests Required.

No definite rule can be laid down as to the number of tests required to determine the exact quality of a lot of material. If the material be steel, of practically one uniform section, and made by a single process, which is reasonably uniform in its operation, one tensile and one bending-test for each melt of steel—that is, for each open-hearth heat or Bessemer blow—is usually considered sufficient. Should the sections vary, or the ingots be of different sizes, requiring more or less work on the steel, each such materially-varying section should have a tensile and a bending-test. Should there be a suspicion that any portion of the material has been burned in re-heating for rolling or hammering, of course extra tests will be required. A provision incorporated in the Standard Bridge-Builders' Specifications by interested parties requires for each contract only four ten-

sile tests, and one additional test for each 50,000 pounds of material, not taking into account the possible non-homogeneity of the lot thus arbitrarily lumped together. But the absurdity of this device for saving labor and expense has been generally recognized, and it is practically a dead-letter in the Standard Bridge-Builders' Specifications to-day.

The advisability of keeping melt-numbers distinct throughout the manufacture of the material, and stamping them upon each of the finished bars, has been the subject of considerable controversy. At most of the structural mills, however, it is now deemed important, not only to the inspector, but to the manufacturer, as facilitating the intelligent selection of material from stock on hand, that not only the original ingots, but also the blooms and finished material rolled therefrom, should be stamped with the melt-numbers, and that the records of each melt should be kept so clearly that all required data of quality can be readily ascertained at any subsequent period.

Good open-hearth or Bessemer steel is reasonably homogeneous as cast into ingots in each melt, but the quality of one melt by no means insures that of the next. It follows that the only satisfactory method of testing finished structural open-hearth or Bessemer steel (short of the separate testing of every finished piece singly, which is, as a general rule, impracticable) must be based on the melts as the units of quantity, and hence that these must be kept separately traceable with the utmost certainty.

The advantages of this system should be at once apparent to manufacturers; yet, strange to say, it has been adopted tardily and with reluctance in many works.

The number of tests required for wrought-iron is still more difficult to fix. Not only does the raw material vary greatly in quality, but the process of manufacture without fusion does not offer any guaranty of ultimate approximate homogeneity. An inspector acquainted—as he ought to be—with the character of the stock used in making the iron, the size and make-up of the pile from which the iron is rolled, and the methods of manipulation employed, can often form an intelligent judgment of the metal and dispense with many tests which he would otherwise require. In general, the usual practice is to obtain one tensile, one bending-, and one nicked bending-test of each materially variant section, mixture of stock, and character of pile in each lot of material or each day's work of the mills. Such tests will average perhaps five or six to the carload, the number varying considerably, according to circumstances.

Even though the inspector be acquainted with the work and satisfied as to the quality and uniformity of the metal, a certain number of tests for each job is required as a matter of record by the engineer for whom the work is done. Such a record is valuable, not only in case of an accident to the structure in which the material is ultimately placed, but also in questions of repairs or of extra-loading. In such cases it is essential to have exact data regarding the quality of the material, as developed by the inspector's tests. Many parties also insist upon such records for personal vindication of the engineer or the corporation he represents as to the care that has been taken to see that all has been done that could be done to insure good quality in the material furnished.

III. In what Cases more than One Test should be Made or Allowed Before Decision is Reached.

This subject has often occasioned much controversy at the mills. It calls for tact as well as good judgment on the part of the inspector, who, if he cannot keep both the respect and the good will of the mill management, has largely lost his usefulness there. Manufacturers claim that where the test-specimen does not fairly represent the material it is unjust not to allow a second trial upon a fairer specimen. On the other hand, some engineers instruct their inspectors that, as the selection and preparation of the test-specimen rests mainly with the mill-people, the failure to obtain a fair specimen is due to their own carelessness or misfortune, and second tests are not to be permitted. The objection to more than one test, if the first fails, is the difficulty of deciding which represents the material fairly, or of finding and rejecting that part of the lot which the faulty test did represent, and of being sure that there is not more of the same class of faulty material in the lot, which will be passed by undiscovered.

The first test sometimes fails, manifestly because the specimen has been burned or has been cut so close to the crop-end as to include a large flaw; or it has been unduly hardened by water on being suddenly plunged into snow or ice in winter, while hot from the rolls, as sometimes happens to the ends of bars by careless handling. In such cases it is difficult to decide wisely and justly to both interests. At mills where much trouble of this sort occurs fast and careless work is being done, and the reputation for this bad work soon becomes known, much to the detriment of the proprietors, who can only regain their prestige by a radical change in their methods,

which usually includes changes in the mill-management. At the same time, a certain amount of it will occur at the best-regulated mills. In the practice of the writer, when additional tests are allowed, either the first test-specimen must have been manifestly, in the judgment of the inspector, erroneously selected, or else more than one satisfactory additional test should be made to counterbalance the evidence of the first and to prove that the material is all right. It makes a great deal of difference, of course, what the result of the defective test indicates. If it shows badly-burned material or non-homogeneous steel with high manganese (from the ferro-manganese not being properly mixed in it), or brittleness from almost any cause, it is only in exceptional cases, under specially peculiar conditions, that second tests should be allowed.

The general rule may be laid down that if more than one test is allowed, where the first shows metal that should be rejected, it should be only in cases where the defective material represented by the first test can be identified, and where a further series of tests will prove with reasonable certainty that there is no more of the defective metal in the lot represented.

Of course, if there has been any probability of error in making the test itself, an honest inspector will make a second one, being careful to note all particulars of the verifying test, including the means taken to secure for it a sample as nearly as possible similar in all respects to the first.

IV. The Way in which Tests Should be Made.

Tests should be conducted with a spirit of fairness to all concerned, and with the object of determining the quality of the material in the prescribed particulars, which have been specified only to secure the rejection of material unfit for the purpose for which the metal has been ordered.

With the newer forms of automatic power testing-machines now largely used, there is very little opportunity to change the results as recorded by the machines. If the specimens themselves are properly prepared and placed in line in the testing-machines, the chances of error are few, and more nearly concern the manufacturer than the engineer. Manufacturers have learned that improperly-prepared specimens and carelessness on the part of those who work and keep their testing-machines in order, operate most severely against their own interests. It is the present experience of the writer that there

is little cause for complaint at the structural mills throughout the country as to the way in which the tests are made.

Kirkaldy strongly recommends determining the tensile strength per square inch of the *fractured area,* rather than of the original area, as is usually done. He claims for his proposal the advantage of stating the tenacity of the specimen in measures of the ductility of the metal as well, which of itself is a good argument. The matter involves several difficulties, however, beyond that of changing the custom already firmly established. To express the elastic limit in terms of the fractured area would have the disadvantage of not actually showing all the facts regarding the specimen, while to calculate the elastic limit upon the original area and the ultimate strength upon the fractured area would lead to much confusion. Again, a practical difficulty is encountered in accurately measuring the fractured area for this purpose, especially in the case of wrought-iron, the metal being liable to open up somewhat at the welds in fracture. Again, specimens sometimes break, not at the point of greatest reduction of area.

The disadvantages of the system seem to more than counterbalance any possible advantage.

V. The Interpretation of the Tests and Test-Requirements.

It is unreasonable to require the same results from tests of iron or steel made on varying sections, even when the method of manufacture and the general arrangements for producing the crude material are the same. For instance, the same tensile tests should not be required from plate rolled from a given grade, size and shape of ingots into $1\frac{1}{4}$ inch thickness, $\frac{5}{8}$ inch thickness, and $\frac{1}{4}$ inch thickness, for the reason that the $\frac{1}{4}$-inch plate will certainly show evidence of cold-rolling, and the $1\frac{1}{4}$-inch plate evidence of not having had sufficient rolling at the low temperature which gives advantageous work to the metal.

It may be remarked here that the use of a small amount of aluminum in the steel will cause the results in these thick specimens to be much more satisfactory, approaching those of the $\frac{5}{8}$-inch plate. Especially is this the case if some care is taken that, in rolling the thicker specimens, the finishing passes are not made at too high a temperature.

Test-specimens cut transversely to the longer axis of the material rarely give as good results as specimens cut longitudinally. Espe-

cially is this true of iron, which shows a marked decrease in ductility and a very considerable decrease in tensile strength in specimens cut transversely.

Few rejections are made upon the requirement for elastic limit in iron or steel. The usual requirement that the elastic limit for iron shall be 26,000 pounds per square inch, and for steel that it shall be at least one-half the ultimate, is very rarely unfulfilled. The elastic limit is not always a safe criterion of the quality of iron or steel. Many tensile tests show a satisfactory elastic limit that are very low in ductility.

In tests of boiler-plate the record of the elastic limit is rarely taken, although this seems to the writer to be an unwise omission. It would be advantageous to have the determination of the elastic limit, and, with the methods of testing usually adopted, it could be approximately obtained with very little trouble and no extra expense.

The elongation in 8 inches length is the almost universal standard. It was adopted by the English Admiralty as being nearest to the 200-millimeter length of the French. Kirkaldy still prefers the 10-inch length. The disadvantage of this length is that it means at least 2 inches more of length of bars or plates from which test-specimens are cut—a matter of no small moment in the large number of tests required in daily practice in structural mills. That a length of at least 8 inches is advisable is shown by the increased amount of irregularity in the results obtained with shorter lengths, the irregularities of the elongation and the method of obtaining it at the fracture very considerably increasing the errors in measurements. The old Government marine test-specimen of 1-inch section is now admitted by all to be a manifest error.

The reduction of area in iron is a requirement which ought not, I think, to be very minutely interpreted, on account of the difficulty of measuring it accurately, due to the incipient opening-up of the welds at the point of fracture. In steel the reduction of area is often the best indicator of the ductility of the specimen, and taken together with the character of fracture, which should always be noted, it is an important factor in the estimation of quality in this respect. The specified percentage of reduction of area is almost always obtained in the best practice, in iron as well as steel, and the difficulty with iron is not that the substantial requirement has not been reached, or even exceeded, but that the exact measurement is not easy, and if it is reported with professed accuracy for the satisfaction of the engineer, he must not infer too much from the figures.

No general rule can be stated for the demand as to the character of fracture under tensile tests, but it may be safely said that for both iron and steel of 70,000 pounds per square inch tensile strength the fracture should show the flow of the metal—fibrous, not crystalline—to at least 80 per cent. in the case of wrought-iron, and that in steel it should be silky rather than spongy and granular.

In the interpretation of tensile tests of steel the hardness of the metal is an important element. It has been found by experience that bridge-steel of over 64,000 pounds per square inch tensile strength is so hard that in punching it minute fractures will be made in the metal surrounding the holes, which will be so injured by the operation that tensile specimens taken from punched pieces show very considerable decrease in tensile strength.

The usual instruction given by engineers to inspectors is that steel which is above 62,000 or 63,000 lbs. per square inch tensile strength should be held quite rigidly to the results of the tests in acceptance or rejection; but where the steel is milder, a test varying 1000 or 2000 pounds over or under the requirements of the specifications is not generally considered, by itself, a cause for rejection. In this connection it is well to point out that duplicate results taken upon the same lot of homogeneous test-specimens, as shown in the tabulated reports of the committee of the American Society of Mechanical Engineers, exhibit a variation much greater than this limit, which is usually allowed in the acceptance of material. There are some engineers, however, who are very rigid in their requirements that all the material shall answer precisely all the tensile requirements. In such cases either the requirements should be very broad and lenient or the metal should be paid for at a considerably increased price per pound.

With wrought-iron the general practice of engineers is to allow their inspectors a considerably larger variation in acceptance of tensile results than with steel. This practice, I think, is warranted. Wrought-iron made in different districts of the country, from varying stocks, gives varying results. In Eastern Pennsylvania, where the furnaces have largely run in the past on foreign ores, the bar-iron is somewhat lower in tensile strength, but is greater in elongation and reduction of area than that made in the western part of the State, from Lake Superior ores. In the eastern district it is easier to obtain the best bar-iron with 48,000 pounds per square inch tensile strength and 20 per cent. elongation in 8 inches, while in the western part of the State 50,000 to 51,000 pounds per square inch

tensile strength is exhibited with 16 to 18 per cent. elongation in 8 inches. In either case the engineer can, in my judgment, safely trust the bar-iron with the same unit-strains.

With a specification for wrought-iron similar to that outlined in our paper upon "The Inspection of Materials of Construction in the United States" (*Trans.*, xix., 911), I believe a reasonable allowance of a few thousand pounds per square inch, or a few units per cent. in ductility, according to the character of the metal being made at the mill, should be permitted to the discretion of the inspector, or, preferably, the specifications should be broad enough to admit both varieties of metal.

The requirement of 52,000 pounds per square inch tensile strength for bar-iron for structural purposes cannot, according to my experience, be met without many rejections by mills in general. Very few mills in this country, except those working charcoal-iron or using steel-scrap in the piles, could produce regularly such bar-iron. An iron having this high tensile strength should be carefully tested for welding, as it is very apt to be dry and incapable of welding strongly, and if it satisfactorily answers both requirements, it should command a higher price than ordinary good bar-iron.

Since the price of soft steel (of not too rigid requirements) and that of wrought-iron are now about the same, it is becoming a common practice to allow in specifications for structures the use of either wrought-iron or steel, giving specifications for both. This practice has led, both at the bridge shops and at the mills, to the interpretation that the material may be furnished in both iron and soft steel up to 60,000 or 64,000 pounds tensile strength per square inch, as it pleases their convenience. The writer has had several experiences of late where the mill people, under such specifications, have actually attempted to fill items of angles and plates partly in iron and partly in steel. This indiscriminate use of iron and steel, with their different moduli of elasticity and expansion and contraction under varying temperatures, does not seem to be good engineering practice. Of course, this remark does not apply to all cases, but only to those in which the differing behavior of the two materials may produce injurious strains.

VI. What Material the Tests Should Cover.

As before explained, the series of tests with open-hearth or Bessemer steel usually cover each melt, and with iron each lot of varying section; but where some of the tests prove defective, they

may fairly be construed to cover smaller subdivisions under certain conditions. If it can be shown that the cause of the trouble is with only one reheating furnace-charge, or even with certain ingots or blooms near bridge-walls or exposed to special punishment, it is manifestly fair to all interests to reject only the material that can be proved to be defective, taking further tests of the remainder of the lot.

VII. The Jurisdiction of the Inspector over Rejected Material.

In one view of the subject the inspector, who, ordinarily, does not own any of the material (although the mill-people claim that inspectors sometimes act as if they did, with the addition of the mill and all the property and people pertaining thereto), has nothing further to do with material that he rejects. Certainly he has no right, without the mill-manager's consent, to maltreat rejected bars so as to prevent their being used for some other purpose. At the same time it seems to me that he has the right to mark rejected material in such a way that it may not, by any accident or otherwise, be shipped forward from the mills with the accepted material. For this purpose many inspectors have a private stamp for stamping rejected material, which seems to be a wise precaution.

VIII. The Jurisdiction of the Inspector over Accepted Material.

The inspector should be furnished with lists of the material he has inspected or shipped on each car-load, in order that he or his assistant, the inspector at the shop, may identify the material which he has passed as it is unloaded at its destination. For this purpose many engineers—it seems to me very properly—insist that their inspectors shall approve the itemized bills of lading of each car-load of accepted material from the mills.

The inspector should, where possible, see that material is properly loaded on cars for shipment, in order to prevent its being badly bent and twisted in loading or in transit. This is a matter of no little importance with the large bars of 60 and 70 feet now often ordered at the mills.

IX. The Right and Duty of the Inspector to Note and Report upon Methods of Manufacture.

Many mill-people claim that if the inspector is furnished with material answering the requirements of the specifications, there their

duty ends, and any investigation into the methods of manufacture is unwarranted and unnecessary and is prohibited. On the other hand, the manufacturers are constantly pressing for fewer tests upon the tonnage they turn out, and in the judgment of the writer the best way to reduce the number of such tests is to make the inspector familiar with the methods by which his material is made, and thus able to judge more accurately, by his knowledge, of the uniform character of the stock used, the size of the ingots or piles or blooms from which finished material is rolled, and the manner in which the material is rolled or hammered. Without this knowledge, and actual testing of the materials used, steel scrap ought not to be allowed to be piled with iron.

Discussion.

W. H. Morris, Pottstown, Pa.: It would perhaps be well to reflect a little on what has been accomplished since the establishment of these "testing-bureaus," and why they should now wish to change the method of testing, and the specifications which they themselves forced the mills to accept. Some years ago, it was the custom to buy a good grade of puddled-iron or wrought-iron made from scrap by standard mills; and my company frequently made, for use in construction, without any machine-tests, plates which were always satisfactory to our customers. Of course, we ourselves inspected these plates, as we still continue to do. With the more general introduction of testing, it was thought necessary to have outside parties make the tests, especially as many mills at that time had no testing-machines. This practice grew easily into a further inspection for surface-defects; and as it is quite human to magnify one's services in business, especially when one's pocket is at stake, requirements were piled one on top of another, and the corps of inspectors was largely increased. The more the inspecting to be done, the more help was required to do it, and the more profit to the testing-laboratories. This development has worked serious injustice to the mills. They are required to handle their iron a second time, and sometimes oftener, because it does not suit the inspectors to be on hand when the iron is made. Sometimes the mills are kept waiting as long as a week, much to the detriment of the customer, especially on hurried work.

Some inspectors, when they begin, know nothing about the manufacture of iron or steel; and some are young men, sent from technical schools to be really educated at the expense of the mills. Worse

still, the inspector has now ceased to make a final decision on the acceptance of material, as was the case when the business was first started, and the engineers claim the right to reject material at any time, even after it has been spoiled by punching and riveting, and is only fit for scrap. In the handling of a large business, it is necessary to load direct from the shearing-bed; and for bulky articles, such as plate-iron, this is particularly desirable, because the re-handling of such material is very expensive. Our plates run up to 8 feet wide and 75 feet long, and we have rolled them over 100 feet long.

Again, with regard to the number of tests, more and more are called for, without any appreciable gain to the consumer. When the mill is using a certain reasonably uniform kind of stock, one or two tests a day should certainly be a sufficient indication of the character of the product, and this ought to satisfy any reasonable buyer. The notion, sometimes set up, of making one or more tests on every car-load, is absurd. Such a requirement shows that people have either no confidence in the party they deal with, or no confidence in the uniformity of the material they buy. In that case, they might as well make a test of every piece. Again, the surface-inspection should be made by the mills, and they should be relieved of any annoyance in this part of the work, at least.

Steel is recognized to be a better material for construction than iron, while it may be generally admitted that, from lack of experience, we are not so familiar with its characteristics. Why should the tests for steel be so much more rigid than for iron destined for the same purpose? In other words, Why should the minimum and maximum tensile strength be fixed within such narrow limits, even to 2500 pounds variation above or below the mean for steel, when no such thing is thought of for iron? If an engineer calculates his strain at 60,000 pounds, why should he find fault if he gets a somewhat stronger piece of steel, provided the other qualities are up to standard? If the strains are calculated on 60,000 pounds, why should fault be found if the material shows not more than 70,000, and gives at the same time the full elongation or reduction required?

Cold-rolling, which Mr. Hunt criticizes, was brought about by these very requirements, it having been found that there is in steel, as indeed in other metals, a certain variation in physical qualities for the same chemical analysis. Does not the same statement apply to iron? When the government will license a boiler for use on the

high seas, merely proportionating the reduction and the tensile strength, why should more rigid requirements be made for bridge-steel, except that the bridge-inspectors think it will make more work for them, or that some engineer has a hobby of his own?

With regard to the relative merits of open-hearth and Bessemer steel, the regularity of product is perhaps in favor of Bessemer, where it is confined strictly to one class of raw material. In the open-hearth practice, the time for tapping is judged by the melter, and hence necessarily varies according to his judgment and that of his partner on the other turn. In the case of a number of furnaces, there would be still greater variation. Delays in tapping present another uncertain contingency. In the Bessemer process, running on one grade of metal, with a constant mixture, and a regular measured quantity of blast, there is less room for irregularity; and any slight changes can be controlled by taking the tests from the next previous heat as a guide. In actual practice, it has been demonstrated that, in this process, variations can be controlled in very narrow limits, and misfits kept below one per cent.

Something is said by Mr. Hunt implying reflection on the honesty of the manufacturers in dealing with the inspectors and specifications. It seems to me that such suspicions are largely brought about by the inspectors themselves. Some inspectors come to a mill with the idea that they are hired as spies, and naturally feel aggrieved if they do not find something irregular to report. When a man is disposed to do what is right and reasonable, and has had any practical experience, I imagine he has little to complain of; but in many cases the inspectors are persons held to rigid instructions, with no judgment, or no right to use it, who report to some theoretical assistants; and of course there is bound to be a "hitch." No man is fit to have charge of inspection and testing who has not had some experience as a manufacturer. I have known of an instance in which an assistant engineer came to the works with the expressed determination of showing that the iron was bad, and had it put to the most unreasonable strains, such as plunging into water while hot after being bent on itself flat, then opened up again cold, to try and separate the welds, and then nicking and breaking the piece for the same purpose. He felt very angry when he failed to prove that the material was bad, as he had predicted. I have also known a case in which an engineer took the position that he could inspect the material or not as he chose; that he could decide that the material was not fit to use, without even taking the trouble of looking at

it; and he even went so far as to attempt to condemn it in that way.

The effect on the mind of a young inspector, coming to a large works where many hands are employed, and finding, after he gets well seated in his position, that he can at will condemn a plate for a trifling defect, is very bad. He naturally feels, like Selkirk, that he is "monarch of all he surveys." Some inspectors have complained, on the other hand, that the mill-men considered them "poor cattle." All such collisions are, however, in my experience, fortunately rare. They would probably never occur if the business were confided exclusively to practical men. A blacksmith who has outlived his ability to do heavy work would make a much better inspector than a green college-boy.

It would be well if some specified thickness of metal could be established for test-pieces, especially in steel; and this, I think, is the custom in England as most soft blooms are sold on a guarantee of test at a $\frac{7}{16}$ inch thickness. The U. S. Government also recognizes, in its requirements for the boilers and hulls of vessels, the difference in materials at varying thicknesses.

It is the custom of the Pottstown Iron Company to make tests of its material for its own information; and every heat of steel has, in addition to four analytical tests, five mechanical ones. There is cast, for each heat, a 4-inch square test-ingot, which is hammered down to $1\frac{1}{4}$ inches square. This is nicked and bent cold, to see how many blows it will stand, both chilled and unchilled. Another piece is then hammered at the same heat to $\frac{5}{8}$-inch square and bent both chilled and unchilled, each piece being nicked and broken, to show the fracture. Finally, a $\frac{5}{8}$-inch piece is pointed, flattened out to $\frac{1}{8}$-inch thick, split open from the point about $1\frac{1}{2}$ inches, and two $\frac{3}{4}$-inch holes are punched, the first being $\frac{1}{4}$-inch from the end of the split and the second, $\frac{1}{4}$-inch from the first, and directly behind it. All this (which may be called preliminary work) is done at one heat, without much care to keep the conditions uniform. The remainder of the work, which forms the real test, is all done at a bright cherry-red, and great care is taken to keep the conditions the same for all tests. The ends of the split are bent down sidewise until the ends of the two parts touch the sides of the main test-piece. Finally, the $\frac{3}{4}$-inch holes are drifted out to $1\frac{1}{2}$ inches. The outer edges of the test-piece, the division between the holes, and the base of the split, should be perfectly smooth and free from checks or flaws. This test, generally known as the "ear-test," forms an almost perfect measure

of red-shortness, or shows the entire absence of that quality. All our material will stand these tests; and, in addition, the plates are tested for tensile strength in the mill, after being rolled.

I have a full set of the tests above named on the table, which I shall be glad to have any one interested examine. The analysis of the blow is: carbon 0.13, phosphorus 0.10 and manganese 0.255. I have also samples of tensile tests, both Bessemer and open-hearth, which show as follows:

	Bessemer.	Open-hearth.
	lbs. per sq. inch.	
Elastic limit,	31,340	33,330
Ultimate strength,	56,520	57,790
	Per cent.	
Elongation,	28.25	29.25
Reduction,	55.70	60.50

Also, some bending-, punching- and drifting-tests; the latter, drifted cold from a $1\frac{1}{4}$-inch to a $2\frac{1}{2}$-inch hole. Also, some nails made from steel (with heads flattened that do not fly off) and from steel scrap, to show the welding-properties. Particular attention is called to the piece $1\frac{1}{8}$ inches wide by $\frac{3}{4}$-inch thick, punched with $\frac{3}{4}$-inch holes, which, under this pressure, expanded the piece to $1\frac{1}{4}$ inches wide at the holes, without a crack.

The mere physical tests, as indicated by a testing-machine, are not nearly so valuable as tests involving such exhaustive strains as these. One test of this kind for each cast of steel should be ample. Under Mr. Hunt's plan for tests, the ordinary output of a mill would involve 50 or 60 test-pieces daily, which would be highly unreasonable.

If the different testing-bureaus would agree on one man for a given works, much trouble would be saved to the mills and to themselves. Often five or six men come to test and inspect severally, and at the same time, relatively small shipments; and the cost of their pay and travelling expenses is a heavy burden for the consumer.

There is no reason that the chemical requirements for basic steel should be more rigid than for acid. The former is a softer metal, and better adapted for uses formerly served by wrought-iron. In heating it, any workman with experience in soft steel can do as well as on iron, if not better. Nothing is to be feared on this score. Soft steel slabs, made by our special processes, have been repeatedly heated as hot as iron piles without any deleterious effects. We have also

piled and welded the scrap from such plates, and cut it up into nails, without discovering any signs of the welds. In fact, our steel scrap can be welded as easily as iron; and the samples here shown of welded steel nails, solid steel nails, and soft steel, speak for themselves. We can get our phosphorus down below 0.01 and keep our manganese low, without any signs of red-shortness. We can make our tensile strength as low as 50,000 pounds. In short, we can make a metal as soft as lead, and from this to any required degree of hardness.

As a final comment on Mr. Hunt's paper, I would suggest the following principles to consumers and their inspectors:

1. Employ ordinary good business judgment in dealing with manufacturers in whom you have confidence; and this will secure, in return, their respect and confidence towards your inspectors.

2. When full tests are made regularly by the manufacturers, accept their tests in lieu of an excessive number of your own.

3. Allow the mills to make their own surface-inspection, or else let your inspectors' acceptance of materials be final, as it used to be.

4. Always select as inspector a man who has had experience in making and working iron.

5. As far as possible, avoid the duplication of tests, and save expense for your customers and yourselves.

6. Adopt some standard thickness, as well as width, for your test-pieces.

7. Make your specification as simple as possible.

8. Allow a lee-way of 5000 pounds above or below the ultimate strength required, or else follow the Government requirements in accepting higher tensile strength, if the elongation shows corresponding increase.

9. Accept any material which is equally as good for your purpose as that which the specifications call for.

Wm. Kent, New York City: As Mr. Morris has said, the mill-men should be heard on this question. The American Society of Mechanical Engineers and the Society of Civil Engineers have committees on the subject of standard specifications and standard tests of material; and if the mill-men do not speak out, it is likely that the reports of the committees to these societies, representing the users of material and consulting engineers rather than the producers, may be too severe in the number of specifications, too rigid in the requirements, and perhaps too theoretical, to suit the ideas of manufacturers. We need a full interchange of views on this subject, so

that the specifications finally adopted in this country may be such as the mill-men themselves will feel to be just and right. There is now a good deal of friction between the customer on the one hand and the manufacturer on the other; and the inspector in the middle is being ground between the upper and the nether millstone. For the sake of all parties, I hope that the question will be thoroughly considered before any definite report is made by the committees referred to. There is no better place for such a free discussion than this Institute, which comprises, to a larger extent than any other body, representatives of both classes concerned. It seems to me that both Mr. Hunt and Mr. Morris are too indefinite. For useful and pertinent discussion, I should like to have each of them, and also every other writer on this subject, codify the specifications he proposes, and not merely tell us in general terms what considerations or principles he advocates and emphasizes. As an appendix to each criticism or suggestion from either side, a definite form of proposed specification, including standards and methods of testing, would greatly assist us in reaching, at last, a satisfactory form.

B. E. FERNOW, Washington, D. C.: Being interested (though in another line) in the question of standard tests, I am surprised to find that there is, as yet, no accepted standard of dimensions for tests of iron and steel. In the tests of timber we have such established standards, and without them it would be impossible to make trustworthy comparisons of reported observations. One of the advantages resulting from the wide range of professions covered by this Institute is (as I have had occasion to learn by experience), that suggestions proceeding from practice in one branch may prove highly helpful to practitioners in another. In this spirit I venture to suggest, as a consoling possibility, at least, that further study may lead, in the testing of structural iron and steel, to a simplification analogous to that which we have reached in the testing of timber. We have found, namely, that the specific-gravity test really expresses in many respects the quality of the timber, thus rendering many other tests unnecessary. While this particular character may not be equally significant for wrought-iron, or even for the more nearly homogeneous steel, it strikes me that, among the various tests considered in Mr. Hunt's paper, or else outside of that list altogether, one or more might be found, which, being fully interpreted, would tell more than they are now supposed to do, and would thus supersede a part of the catalogue now apparently deemed necessary.

OLIVER WILLIAMS, Catasauqua, Pa.: Inspectors, as a class, have

greatly improved during the last few years. When the railroad-companies did their own inspection, they were in the habit of sending for that purpose young men fresh from technical schools, whose ignorance of their business was only excelled by their assumption of knowledge, and by their earnest effort to condemn everything they could by any possibility take exception to. This has been changed; and now railroads that do their own inspecting, as well as the inspecting bureaus, generally send men of experience and good judgment, with whom, as a rule, it is a pleasure to do business.

One of the principal present difficulties is, that there is no generally settled specification of requirements. Last week we had in our mills, at the same time, three or four inspectors, representing different bureaus, each one of which had its own special requirements, varying for plate-iron from 46,000 up to 52,000 pounds tensile strength, and from 5 up to 15 per cent. elongation. As Mr. Morris well observes, one inspector could have attended to all these orders at a considerable saving of expense, and with comfort to himself as well as to the manufacturer.

Another point is, that some railroad- and bridge-companies have raised their requirements, and yet are not willing to pay the necessary price to obtain a high class of material. It is a general opinion that iron is not produced to-day of as good quality as fifteen or twenty years ago; but I claim that never in the history of the trade were mills making as good iron as they are generally producing to-day. There is now no difficulty in getting an iron that will meet the most rigid requirements, if the consumer is willing to pay for it. They call for "double-refined" iron, for instance; and there are mills to be found (although they are few) which will take old rails, work them down twice, and stamp the product "double-refined." This, of course, can be done at comparatively small cost; but honestly-made "double-refined" iron cannot be made for the price that is now being paid by the bridge-companies.

The remedy for present evils, as all admit, is a reasonable and uniform standard specification for both steel and iron. I hope this will be reached through discussion, and through the co-operation of this Institute with other technical societies. Mr. Kent asks what the standard specification should be. As a manufacturer, I would say that the Bridge-Builders' Standard Specification meets my views. There are, no doubt, some requirements in it that are too low to suit many of our engineers, but these can readily be modified and, in some instances, raised, to suit their ideas. Take it all in all, it is, in my judgment, a specification that we can meet, and meet honestly.

Some specifications come to us, more especially from the West, asking for iron of very high tensile strength, say 52,000 pounds and over, and no other requirements named. Every manufacturer knows that this single requirement can be readily met, and yet the iron may be practically worthless. When the railroad-companies will employ engineers who are well-posted in iron- and steel-making, specifications will be furnished that can be met fairly and fully, and a material will be supplied that can be depended upon every time.

I trust the discussion of this subject will result in doing away with some of the annoyances and unnecessary expenses to which mills are now subject. The railroad people may think that it is the manufacturer who is paying these unnecessary bills, but in the end they come back on the consumer. I give away no trade-secret in saying that.

I wish to emphasize my concurrence with the point made by Mr. Morris concerning the inspection that does not inspect. It is becoming a custom not to consider the inspection at the mill as final; and after material has been once accepted, it is frequently rejected in the bridge-shops. This will inevitably lead to trouble and litigation in the near future; and it is very important that settled rules governing this final rejection should be agreed upon by the parties in interest.

In connection with my observation that consumers are often not willing to pay for the special qualities they require, I will give an instance from my own experience. A few years ago I hit upon a mixture of metal that made an admirable railway-axle. Dr. Dudley will admit that there is nothing more important in railroading than that. The requirements of a certain railroad-company were that this axle-material should stand two blows of 1640 pounds falling 10 feet, and three blows of the same weight falling 15 feet. I offered to guarantee for my metal double the weight, double the height of fall, and double the number of blows, for one cent extra per pound; but the offer was refused.

C. B. DUDLEY, Altoona, Pa.: As one who has possibly wasted some ink in trying to write specifications, and has had a few contests with the manufacturers of iron and steel, I may be able to throw a little side-light on this discussion. As my friend, Mr. Coxe, just whispered in my ear, the key-note to the situation in inspecting materials is the fact that any inspection is made at all; by which I mean, that if the manufacturer of material knows that the material is going to be subjected to *some* test, whatever it may be, the material

that you get will be better than if there were no testing at all. This remark involves no reflection on the business integrity of manufacturers. It expresses simply our understanding of ordinary human nature. We all know that everybody does a little better when he is watched. It is this fact, that there is some one on watch, if I may say so without accusation of conceit, that gives a portion of its value to the testing-department of the Pennsylvania railroad, with which I am connected.

Some years ago, as many will remember, there was a little contest over the quality of steel to be used in rails. At the beginning of that contest, a good friend of mine, who is a very able manager and at the head of an establishment which has been a perfect marvel of business success, remarked, substantially, "What is the use of our paying any attention whatever to quality? We can sell more rails now than we can make." That was one side of the question—the simple business proposition of getting rid of the output of a mill, no matter what it turned out, and of making money out of it. This, you will understand, was twelve or fifteen years ago. I do not think you will find that spirit now. Let me give you the other side of the question. I have seen in Altoona a large pile of broken spiral springs taken out from the cars, axles broken in service, and cast-iron wheels broken in service, broken crank-pins, and other parts of locomotives or cars which had failed in service. We who are connected with the use of these materials have to meet some pretty sharp and bitter problems. Take the crank-pin, for example. A set of fifty crank-pins is ordered; they come for examination; we select a pin, cut it up and test it. Recently we had a case of this kind. One of the test-pieces cut out of one side of a pin gave 88,000 pounds tensile strength, and 22 per cent. elongation. Another piece from the other side of the pin had 106,000 pounds tensile strength and 14 per cent. elongation. The question is, Shall we put that material into service? I say to you that mechanical engineers who have to face such a problem as that, and who know the effect of putting such steel into such an important thing as a crank-pin, must hesitate. Evidently the difficulty with the metal is lack of uniformity. Both samples are good metal; but is it safe to put a pin made up of such irregular material into service? I could multiply these cases indefinitely from the records of the ten or fifteen years' experience I have had. Now, between the perfectly legitimate and proper desire of the manufacturer to make money by turning out as much material as he can, and getting all he can out of his plant, and the

fact that the man who uses the metal has got to take risks in connection with it, the real conflict comes. I am not very familiar with specifications for the coarser forms of construction along the track, such as bridges; but with regard to the requirements that are put into the specifications, I think there is in the minds of manufacturers a very great mistake. It is a common opinion, I think, among manufacturers who have to furnish material, and among what we call "supply-men," that specifications are almost all written "out of a man's head." That is not true, as I know; for I have written a good many of them. I do not sit down to write a specification and call for what I think would be ideal material. I have never done anything of that kind, and I am the less inclined to do it the older I get. I know that there is a possible tendency on the part of a man fresh from school to think that he knows it all; we have all been through it; but such a man is, of course, hardly fit to write a specification. The specifications that issue from the motive-power department of the Pennsylvania railroad have a tangible and sensible reason for every requirement they contain. If you were in the same place, and were making those specifications, you would put those same requirements in. That is a pretty strong statement; but I say frankly, there is a great deal behind these specifications. It is our custom, when we take hold of a new subject, for which a specification is to be ultimately made, to examine first the good and bad material from actual service.

To give a single illustration: we have recently written a specification, which is still under advisement, for boiler-plate for the inside fire-box sheets of locomotive-boilers. Our first step was to collect samples from about sixty worn-out boilers whose life was known. On a piece representing each fire-box, the ordinary physical tests of tensile strength and elongation were made. Then thirty of those test-pieces were turned over to the laboratory and carefully analyzed for carbon, phosphorus, silicon, manganese, sulphur and copper. With these data at hand, we sat down to write our specification. Let it be remembered that a part of these plates were of good steel, that is, steel of long life, and a part poorer; and that all of them had done service more or less. I think you will admit, that with such data a specification could be written intelligently. After the specification was drawn up, it was first sent to the purchasing-agents, with the request that they would send it to all the manufacturers of that material from whom they might possibly wish to buy, and obtain their criticisms. This is our universal custom. We never issue

a specification without consulting the manufacturers. The specification is likewise sent to the mechanicrl officials of the Company throughout the road, for their criticisms. When all these criticisms have been collected, everything is embodied in the specifications that can be deduced from the experience on both sides. I will say frankly that this proposed specification for boiler-plate has been under discussion for a number of months, and we have not yet issued it. We spent two days recently in Pittsburgh at one of the large steel-works, talking over all possible requirements, ascertaining whether they could meet them, and taking every suggestion that they had to offer. Now we are making a lot of special tests on plates furnished by manufacturers, to see whether we can make certain modifications in the specification, which they think would make it easier for them, and, as they claim, make us a better metal. When all these requirements shall have been adopted and put in print, we shall think we have got a good reason for each and all of them. We know that no human work is perfect; and we generally find, to our annoyance, that with all the care we have taken, we have still to revise our specifications. If the first draft of the specification, thus made with the utmost care, will work in actual use for nine months without running across some difficulty, either in the manufacture or in the service, which will call for further amendment, we shall think we are doing very well. Some of our specifications have been revised four times, and we are constantly revising as we gain new knowledge. I do not say it in any boasting way, but I do really think, if I may judge from my own experience in the matter, that there is a great deal more of meaning and definite reason back of specifications than the mill-men are inclined to admit.

I could go into details as to a good many points, but I will only state a single case. We propose, in our new specifications for boiler-plates, to make what we call a "homogeneity-test." It is entirely a new thing; at least, none of the manufacturers whom we have consulted over the matter were familiar with it; and some refuse to give any opinion upon it. The test is exceedingly simple. We take a piece of the sheet under test, say an inch wide, and 5 or 6 inches long, nick it on one side, put it in a vise, and bend it over with a hammer until we break it. That will give us, as you see, a chance to look at the end; and that slow bending opens out and discloses whether there are any little spaces in there, that were probably originally gas-bubbles, or bits of enclosed slag, which have been flattened out simply in rolling, and not welded up. That is a very common

behavior of steel; and those who have not tried it will be astonished to find how commonly some of the mild steels present that appearance. These little cavities open out beautifully under the test. Some pieces break off short and do not show anything. We regard the cavities disclosed by this test as evidence of the presence in the steel of gas-bubbles which have simply rolled out, or of included slag. We could not ask for any better tensile strength and elongation than we can get out of samples, which will show abundance of these little openings. A practical manufacturer would say, "What do you care whether those little things are there or not?" Let me tell you why we care. The plate is going to be used in a fire-box, on the inside, next to the fire; in other words, a sheet of metal, possibly three-eighths of an inch thick, is to have on one side a temperature of perhaps 3000° F. from the burning coal, and on the other side, where the water is, about 350° to 370° F., depending on the steam-pressure. The heat, passing through from the fire-side to the water-side, comes to one of these little flattened-out bubbles, or layers of enclosed slag, which may be possibly a quarter-inch in diameter; and, since there is a break in the continuity of the metal, the heat does not go as fast there as it does through the contiguous portion. The result is, that that portion of the metal which is on the side towards the fire at that spot, is heated hotter than the contiguous portion, where the heat passes without any break in the continuity of the metal. As every one knows, heat expands metal; and the consequence is that the overheated portion expands so much that it lifts up and becomes what we call "blistered." Now experience shows that, with the possible exception of cracks near stay-bolts, blistering is the principal cause which makes the renewal of fire-boxes necessary. In view of this explanation, we do not hesitate to say that we think there is a good sound reason why we should ask for a steel which is as homogeneous as it is possible for modern methods to make. I could multiply such illustrations.

The question has been asked, why there should be an upper limit fixed for the tensile strength of the metal. We think there is a very good reason why there should be an upper limit. In the matter of bridge-rods we have had bitter experience in the use of "high" steel. There was a certain Howe-truss bridge over a small creek, which it was desired to strengthen, so that it would better sustain the heavier engines and heavier cars which are characteristic of the last ten or fifteen years in railroad practice. Accordingly, from the first two bays out from each pier, an iron rod was taken out, and a steel rod

put in place of it. These rods were 1⅛ inches in diameter. The steel tested 96,000 pounds tensile strength, and 22 per cent. elongation. It was perfectly homogeneous, was made in the open-hearth furnace, and was in reality a magnificent metal. And yet those rods broke, one after another, and we had to keep replacing them. Now the static strain on those rods was but a trifle over 8000 pounds per square inch; and the iron rods in the bays, more toward the center of the span, which were strained 7500 pounds per square inch, static load, and were of the same size, and had only 50,000 pounds tensile strength, did not break. At one time this was a mystery. We made a study of the case, and reached the rather paradoxical conclusion, that if the metal had not been so strong, it would not have broken. I think I can make that clear, and show you why we think an upper limit is of some value. This piece of paper, bent at right angles, may represent the angle-block which rests on the top-chord, and my pencil will represent one of the rods, reaching down through the angle-block to the chord below. Since, as is well known, bridges are not accurately built, and even if they were so, are often distorted by the load, it follows that the nut on the end of the rod through the angle-block often does not rest on the angle-block, over the whole surface of the nut. Actually, in the case which we have under consideration, this was found by examination to be the case: namely, the nuts bore against the angle-block on one side, instead of lying flat on it, so that the whole load was in reality sustained by only a portion of the metal in the rod. This portion that was sustaining the load was unquestionably strained far beyond its tensile strength, and gradually gave way, until finally only a small portion of the total section of the rod was left, and the rod broke off. But you ask, Would not wrought-iron behave in the same way? I answer, No; for the reason that, as is well known, along with high tensile strength in steel comes another quality, namely, great stiffness or unwillingness to bend, a quality which does not characterize iron. Iron rods, under these conditions, yield or bend a little, and give a complete bearing to the nut, while strong steel refuses to bend, and consequently must break. Mild steel behaves very much like iron; and if the engineers, instead of using a metal of 96,000 pounds tensile strength in this case, had bought the metal under specifications that it should not exceed 65,000 pounds tensile strength, it is probable that the rods would not have broken.

I will give you one more illustration, showing why we do not think it is unwise to give an upper limit beyond which your metal

shall not go. Several years ago, we had a specification for boiler-plate which said: "Not below 50,000 pounds tensile strength, and not less than 25 per cent. elongation." This was in the early days of steel boilers. We began to get the metal, and found that some of it had 75,000 pounds tensile strength, while the elongation was all right. You might say, every one of you, that that was absolutely better than we asked for, and that surely we ought not to complain of such metal; yet, when we took it to the boiler-shop, and began "flanging" it, it cracked and went to pieces. So we introduced into our specifications words to this effect: "Tensile strength not less than 50,000 pounds, and not over 65,000." We think that there was really a good, sound, sensible reason why there should be an upper limit in that case.

I am quite ready to admit that there are specifications made that are unreasonably difficult to fill, but I am not at all ready to admit as a deduction that the whole system of specifications and testing should be abolished. The evil is in the unreasonable specifications, and not in the care that those who use metal must take to get such material as will be satisfactory in actual service.

In consultation with Mr. Theo. N. Ely, General Superintendent of Motive Power of the Pennsylvania railroad, by whose authority, and under whose supervision, all the specifications for materials in the motive-power department of that corporation are issued, I have found him strongly in favor of the maintenance of the specifications. Mr. Ely says:

"So long as I have anything to do with the matter, the specifications shall have my full support; and I should feel that we were going back twenty years into the dark ages in the purchase of supplies for all purposes, should we retreat in any degree from the position we have taken in the incorporation of specifications into our system of railroad-management. While we have, as a railroad, gained greatly through their instrumentality, the manufacturers themselves have been benefited in an almost equal degree, in that those who did not really wish to make honest material have been excluded from our patronage, and have not been allowed to undersell the more reliable manufacturers. The Pennsylvania Railroad Company's specifications have been adopted by railroads scattered widely over the country, with the effect of enabling manufacturers to prepare standard stock with reasonable assurance that it could be disposed of. With the routine adopted in the preparation of specifications, which anticipates consultation with the manufacturers, I can conceive of no reasonable ground for complaint, and no method which would produce equally good results. We have on the one hand the users of the material, and on the other hand those that manufacture it, working together to produce the best results. Every intelligent manufacturer must see that it is to his advantage to be in close communication with his customers in order to meet their requirements. It is equally true that the railroad-companies, or other users, must familiarize themselves with dif-

ferent processes and their possibilities, the end to be gained being a material best adapted to the service, at a cost which will not prohibit its use.

"What I have said applies to design as well as material. While a free and open discussion of matters of this kind, conducted without prejudice, results in ultimate good, I would have no patience, and think it a waste of time and force, to discuss the general question as to whether material should or should not be purchased under specifications. By specifications, I mean those that have been prepared with care, and which would be entitled to the respect given to any careful investigation."

ECKLEY B. COXE, Drifton, Pa.: I speak neither as a manufacturer nor as a purchaser of iron, but rather to justify, from my experience in another line, the remark which Dr. Dudley quoted from me as to the wholesome effect of *outside* inspection of *some* kind, whatever the kind may be. As some of you may have heard, there are two things in "coal,"—coal and slate.

A MEMBER: Which do you send to market?

MR. COXE: Both (laughter). Nobody objects to slate in coal,—just as nobody objects to water in brandy, provided there is not too much of it. Now, in order to satisfy our selling department, and, through that department, our customers, I got up an elaborate system of inspection. At first, the department having charge of the preparation of coal made its own inspection, with daily reports covering every car-load shipped. But after two or three years I concluded that most of my time was likely to be spent over disputes between the New York selling-department and the other departments as to whether a certain car-load of coal, shipped perhaps seven months before, had 7 per cent. of slate in it or only 5. Of course, such a dispute was between a sheet of paper on one side and a consumer's complaint on the other, and the result was nothing but irritation. So we simply said to the New York office: "Put your own man here, and instruct him not to bill a car-load of coal unless it satisfies your requirements." That is still our rule. The preparation-department has nothing to do with the inspection. If a car-load of coal is not satisfactory to the agent paid and instructed by the New York sales-department, he refuses to bill it, and it goes back to the breaker. There was a good deal of "kicking" for the first month, but now nobody would willingly go back to the old plan. We hardly know what it is to have a complaint about the coal. The quantity turned back by the inspector is scarcely measurable,—not a hundredth part of 1 per cent. of the total product shipped. Yet the number of complaints used to be so great as to be a daily annoyance. The difference is not due chiefly to improvements in our mechanical system, though improvements have been

made; nor is it merely that unfounded complaints have been cut off. The real reason is, that everybody knows there is an outside inspection. The breaker-boys know that within an hour after the coal has gone out of the breaker, it will come back again if they have not properly attended to their duty, and they will have to stay there after six o'clock to repair their neglect. Every slate-picker, every screen-boss, every platform-boss, every man about the breaker, knows that if coal thus comes back, there will be growling among those who are thus put to extra trouble, and that the blame will be brought home to somebody, not by the manager only, but by everybody concerned. The inspector does not turn back one car in thousands; but the fact that he is *there*, is entirely independent of anybody else who is there, and would just as soon reject a thousand cars as one, has a very salutary effect. If he should prove unreasonable, that would be reported to me, and I should have a conference about him with the New York department. In fact, nothing of that kind has ever occurred; and our experience is perhaps all the more valuable to others as showing that, even in a business, the two departments of which are controlled by ourselves, we find it advisable to separate them entirely in this respect, and to make one department a check on the other, by providing for the inspection of results practically in the interest of the consumer, independent of the producer.

R. W. RAYMOND, New York City: The principle stated by Dr. Dudley and Mr. Coxe is, I think, of universal application, and so important that further illustrations of it, though apparently far-fetched, may serve to emphasize it. To begin with the most remote: the granting of degrees at our institutions of learning, upon examinations conducted by the teachers of the candidates, is like our inspection of iron, made by the manufacturer himself; and the result has been to cheapen the value of all such distinctions. Whenever the principle of a true, impartial inspection is introduced, the value of the result rises. Thus, the degree of M.D., given by the University of London, upon severe examinations only, and to any candidate, wherever he may have studied, is, as I am informed, so highly prized that medical students come from all quarters to strive for it; and whoever wins it and can put the symbol of it upon his card is sure of professional business at once. No doubt many college faculties conscientiously maintain a high standard as to the bestowal of degrees, though it is equally certain that not all of them do so. But the great fact remains, that self-inspection is not conclusive or satisfactory.

Within the limits of one and the same business, it is highly important that both the inspection of product and the auditing of accounts (which is also a kind of inspection) should be separated from the manufacturing department altogether. The larger the business, the more easily this can be done, because on a great scale of operations there will be work enough of this kind to justify the maintenance of separate departments to attend to it. In smaller establishments it is perhaps not so easy; but I am convinced that many concerns which are not considered by their owners to require absolutely independent inspecting officers would find profit in this additional expense. At all events, so far as practicable, the man who is wholly or partly responsible for the quality of a product ought not to pronounce the decisive judgment upon it. It is better for him, as well as for his employers, that his work shall be tested by another.

On the other hand, I heartily concur with the view of Mr. Morris and Mr. Williams that when a purchaser upon prescribed specifications has had the opportunity of testing material through his own inspector, the acceptance of the inspector should be final. There is no justice in a vague and indefinite responsibility. If the buyer proposes to hold the seller responsible for ultimate results in practice, let him say so, and not attempt to prescribe the means. Let him say, I want a rail that will outwear so many thousand tons of traffic at such a speed in such a position on such a track,—and get it, if he can, or refuse to pay for it if he cannot. But if he says, on the contrary, I want a rail of such composition, enduring such and such tests, and I want my own inspector to try it and accept it; then he ought to stand by that bargain, whatever may be the result in practice. The same should be true of all construction-iron or steel. And if this principle were enforced, it is clear that there would be an irresistible pressure upon consumers and their engineers, both to draw specifications with care and to select inspectors whose judgment they could afford to rely upon as final and safe. It may be because the engineer thinks he can rectify by subsequent rejection any carelessness or ignorance of the inspector, that he is willing to let a young fellow, without the necessary experience, do the testing.

One thing more, and it might be made the text for a long discourse. Unnecessarily severe requirements always involve lax enforcement. Reasonable and practicable conditions can be and ought to be strictly enforced. Maximum requirements frighten away sometimes the honest bidders, and invite those who either do not

understand what they are promising, or rely on evading its performance in some unworthy way.

Giving wide discretion to the inspector is not the best remedy for this vice in the specifications. It is not fair to the manufacturer or the inspector that the *letter* of the specifications should be so unreasonable as to require to be set aside. Better demand less, and be more sure that you get it. *We put our severity in the wrong place.*

WALTER E. KOCH, Pittsburgh, Pa.: An experience of twenty years in the open-hearth steel manufacture, and a less extended experience with Bessemer and Clapp-Griffiths steels, has led me to regard the following tests as the best:

1. A straight test-piece, 20 inches long, properly prepared for 8 inches in the middle, should be pulled, and the elastic limit, ultimate strength, elongation, and reduction of area noted. I think 8 inches, and no other length, should be the standard for test. The present 1-inch marine test is absurd and of no value whatever.

2. When plates or bars are annealed, I think tests should be taken after annealing.

3. The drifting-test, if done fairly, and in presence of the maker's representative, is a good one. It has been in use in some places for years.

4. The cold-bends and quench-bends are both essential—especially the latter for steels running above, say, 0.18 in carbon.

5. One test per melting-heat should be the minimum.

6. I would never allow an inspector to stamp rejected material. This was tried once in Scotland, and had to be abandoned.

7. I am strongly in favor of having all inspection done at the mill before shipment. The "coupon" system is unfair. *Audi alteram partem* should be the motto over every testing-room; and surely, both parties should be represented at so important a time. Moreover, much delay and trouble is often occasioned by the coupon-system, to say nothing of accidents caused by the projecting coupon.

8. I think, also, that inspectors should be made conversant with the methods of manufacture. In this way, mutual confidence is established to the great advantage of both parties.

JOHN J. WILLIAMS, Youngstown, Ohio (substance of a communication forwarded to the Secretary by Mr. Hunt): In beginning the manufacture of structural iron we encountered many difficulties and annoyances, connected with the securing of suitable pig-iron, the proper adjustment of rolls, etc., and the training and disciplining of

workmen. After getting, by careful selection of ores, the required quality of pig, we educated our puddlers until they knew how to produce, not merely a sound-looking, but a really good muck-bar, and then held them strictly to the fair requirements, which they clearly understood. This was no small task to be performed upon more than 400 puddlers and helpers. I need not say that to determine the adjustment of grooves and rolls, the size and construction of piles for given sections, etc., so as to secure sound edges and flanges, maximum reduction, proper flow of metal, etc., without sacrificing the physical qualities of the finished shape, was not the work of a day. When all these conditions had been perfectly met, the result might be spoiled by over-heating or under-heating.

Having run the gauntlet of the blast-furnace, the puddling-furnace, the heating-furnace, and the rolls, we encountered the inspector. At first introduction, we felt a good deal of prejudice; and a short experience brought decided ill-feelings. I speak frankly, and for our own case only. Perhaps the fault was partly on our side—but it was not wholly so. We thought that our honest and laborious efforts to overcome such difficulties as I have mentioned, entitled us at least to be judged by men who appreciated them. We felt that we had just cause to complain when (as was sometimes the case) men were sent to us as inspectors who had not the necessary experience to warrant the indiscriminate condemnation of material in which they indulged. I hold the same opinion still. In some instances, we refused to make iron subject to the inspection of particular men, in whom we had no confidence.

Experience soon taught us that suspicion and distrust were mutual. Samples set aside in the inspector's absence were examined and questioned as if suspected of being fraudulent substitutes, not genuine representatives of the iron to be tested. Sections were required to be cut from bars already prepared for shipment; and against such demands, involving loss and delay in filling orders, we naturally "kicked." So the fight went on, until we were about ready to let "some other fellow" have all the pleasure and profit of the structural-iron business. But some of our friends went on sending orders, and took pains to send us more experienced and reasonable inspectors; and, little by little, the situation became better, on both sides, probably. We, perhaps, were toned up a little, and the inspectors were certainly toned down a little. At all events, our intercourse with them has been, for some years, as pleasant as could be wished. We have got beyond the *suspicion-line* on both sides; and

the result of the high standard of excellence earnestly maintained by us, and cordially recognized and assisted by them, is an undoubted benefit to every department of the business.

From my experience I should say, that 50,000 pounds tensile strength, 20 per cent. reduction, 25 per cent. elongation, and 30,000 pounds elastic limit, would be a fair formula for the requirements of structural iron. Higher figures may be reached, with careful selection of material, and continuous vigilance at every stage of manufacture from pig to bar; but it seems to me, sometimes, that those who require so much, in ordinary market-purchases, incur great risks of failure.

To Mr. Hunt's suggestion, that inspectors should be permitted to note and report upon methods of manufacture, including material used, size of piles, manner of rolling, etc., I have always been, and shall, I think, continue to be, decidedly opposed. Whatever may be the competency of individuals, inspectors as a class are certainly not competent to judge of details which it takes a lifetime to learn. They certainly ought to know how to measure the results obtained; but that does not involve knowing all about the means. Moreover, the final judge of results ought not to be also the critic of the means by which they are attained. If he has previously approved the method of manufacture, or, perhaps, has even suggested improvements in it which the management has adopted, he must be prejudiced in favor of the results; and, if he does not like the methods, or has made suggestions which were not accepted, he will be tempted to find in the metal he inspects the faults which he is sure his rejected advice would have prevented. When it becomes necessary for me to permit the process of manufacture to be supervised by inspectors who have the power of finally rejecting the product, while I bear the responsibility of the business, my place will be open for a successor.

I have omitted to mention, as perhaps a matter of some interest, a test practiced in our works, which, I think, is not common, namely, a test of elongation along the whole bar, from grip to grip, in which the reduction is very uniform on both sides, almost to the breaking-point.

GUSTAVUS C. HENNING, New York City:* For the intelligent discussion of this subject it is essential that a complete study of it

* Mr. Henning's contribution was presented at the Baltimore meeting, February, 1892, and the concluding remarks of Mr. Hunt were subsequently received. Both are published in this volume, in order to bring the whole discussion together.

shall precede any deductions. Personal experience may replace study to a certain extent; but in rare cases only is this sufficiently comprehensive to be the basis of sound judgment. A knowledge of the literature relating to it is indispensable; and as much of that literature exists in foreign languages only, it is a task of no mean proportions to become posted in the different branches. The work of an "inspector of materials" is hampered by much clerical work, mill-routine, and other troublesome matters, so that he rarely has an opportunity to do anything but carry out the work directly entrusted to him. It is also true that routine-work and commercial testing rarely develop new methods, these being deduced from experiments and tests carried out in laboratories, where the behavior of materials under different treatment and conditions is investigated by those who have the time and freedom to look at the subject from all sides.

The methods now obtaining in this country were not introduced by mill-managements, bridge-works or inspection-agencies, but gradually by engineers or individual inspectors, and in spite of strenuous objections from the mills and shops. The methods proposed and now quite generally adopted had been, with few exceptions, deduced and proved by laboratories in this country and abroad. Ten years ago very little testing was done, and under widely varying conditions or instructions from engineers; and instead of improving or facilitating or cheapening the work, as done at first, it appears to have been growing poorer and more superficial.

It seems perfectly proper and logical that the party who buys material shall have the right to say which and how many tests shall be applied, and that the mill-managers shall have the right to say how such material shall be made. It would be strange indeed to make the manufacturer the judge for the buyer as well.

In his paper (page 679), Mr. Hunt states three propositions, which are called self-evident, but a study of the second and third of these shows them to be anything but that. In the second it is assumed that testing is a matter arbitrary with the engineer and left to his discretion alone, whereas it is always clearly defined in a properly-written specification, and there is never any trouble when the prescribed requirements are fulfilled. Nor is the expense considerable, except when the material is not satisfactory, and particularly when the contractors have not properly studied the requirements before making their bids. It is well known that certain specifications, which are quite reasonable for wrought-iron made in the Pittsburgh district,

cannot be filled in the East; and that others drawn for Eastern mills, and which would there secure first-class material, would be unsuited in the West to obtain equally satisfactory results. Still, these different specifications are used indiscriminately in the different districts, and prices are made irrespectively. Can it be surprising that trouble results?

The engineer, having written his specifications for the purpose of obtaining a certain kind of iron for the price bid, it seems self-evident that he has the right to call for any number or kind of tests to obtain a complete knowledge of the particular lot of material offered to him at a definite charge for each kind of test, agreed upon in advance, and to be paid by the engineer. Every mill-manager must know the precise cost of all kinds of tests, as thousands of them are made; hence this item of expense should never be in doubt. Furthermore, every department of a mill should be self-sustaining and show a profit at the end of a year; hence, if a given profit be added to the cost of making tests, no complaint can be made as to their number. It is only bad practice to leave so important a matter to arbitrary arrangement, after contracts have been signed. There should also be no trouble in obtaining the material from which to cut test-pieces; for if shapes are properly cropped, and not pared off as scantily as has become the practice in some mills, there will always be sufficient on hand for this purpose. The argument that test-pieces must not be cut from crop-ends because these are not good material, is not a sound one. If the material is unsatisfactory on one side of a shear-cut, it cannot be satisfactory on the other side immediately adjoining. A crop-end should be so long that there can be no doubt about the quality of the remaining material. The argument that it is expensive to have large crop-ends or much scrap is only partially true, because the only loss, as a rule, is not at all the cost of material cut off, but mainly the cost of labor required to reconvert it into a merchantable shape (except in such works as do not make steel, but purchase it elsewhere and merely roll it). We understand well enough that these are favorite arguments in mills; but as we acquire experience in mill-methods, we are better able to appreciate their force. It is, however, probable that in some of our rolling-mills, where, by reason of lack of proper system, the total wastes are greater than the total profit of an English mill, testing and cropping may cause extraordinary expenses.

The third proposition is not more self-evident than the second. Specifications were never written at random or previous to obtaining

an approximate knowledge of the qualities of materials; and it seems odd to read in a paper presented at a meeting of this Institute of "unnecessary tests," "too rigid requirements," and the like. There never has been a specification written for wrought-iron the requirements of which were not easily fulfilled, after a little effort, by the best mills, such as the Union Iron Mills of Carnegie Bros. & Co., and a few others, although material of the required properties could not have been obtained in the East. Quoting from well-known specifications, such as those of Bouscaren, Vaughan, Cooper and the Bridge-builders' so-called "standard specifications," we find the following requirements.

The first (Bouscaren and Vaughan) has been called severe and difficult to fill, and is generally declared to be too rigid, although without reason; the second (Cooper's) is oftener adhered to, and answers well for Pittsburgh iron, but cannot, except in some cases, be met by eastern mills; the third (Bridge-builders') is used more or less, but the trouble with it is that anything which any mill turns out will fill the requirements prescribed; and if the first set of test-pieces are not satisfactory, a set can be obtained under these specifications which will give the desired results for the record. They are also illogical and contradictory in several points. On page 720 I give average results of tests, taken from my own record, of iron from Pittsburgh and eastern mills, which will illustrate these remarks more fully.

These results show that there would be no difficulty in meeting the first three specifications in Pittsburgh mills, but that eastern iron would never answer. On the other hand, while the eastern iron would always more than answer for the bridge-builders' specifications, the poorest iron made in the Pittsburgh district would do the same. Cooper's specifications cannot be met entirely in the East, except when they are interpreted liberally, according to their prefacing motto: "The most perfect system of rules to insure success must be interpreted upon the broad grounds of professional intelligence and common sense." There is no intention on my part to condemn eastern iron. It is really good material; but I desire to call attention to the fact that it is of a totally different kind, and should be recognized as such in contracts and specifications, just as much as charcoal- or Swedish-iron, or high-carbon and low-carbon steels.

Specifications for steel have likewise given no trouble at certain mills, although others could not have filled them. But in both cases bids have been handed in without regard to requirements, and even

Table Showing Test-Requirements under Several Current Specifications.

Qualities.		Bouscaren and Vaughan.	Cooper.	Bridge-Builders' Standard.	
a. Elastic limit		26,000 lbs.	26,000 lbs.	26,000 lbs. for plate; half the tenacity for tension-bars.	
Tensile Strength:					
b. Tests of full size, 4½ in. sq. or under	Tension members.	50,000 lbs.	50,000 lbs.	1 *sq. in. area*, 49,850 lbs.	4½ *sq. in. area*, 46,830 lbs.
c. " larger than 4½ in. square	Tension members.	1000 less per sq. inch, down to 46,000 lbs.	50,000 lbs.	5 *in. area*, 46,670 lbs.	7 x 2 *in. area*, 43,060 lbs.
d. " ½ in. specimens of 4½ in. section	Tension members.	52,000 lbs.	52,000 lbs.	1 *in. area*, 50,250 lbs.	4½ *in. area*, 49,150 lbs.
e. " ½ in. specimens of larger section	Tension members.	500 less per sq. inch, down to 50,000 lbs.	Down to 48,000 lbs.	4½ *in. area*, 49,150 lbs.	7 x 2 *in. area*, 46,550 lbs.
f. " ½ in. specimens of angles and shapes		50,000 lbs.	48,000 lbs.	47,165 lbs.	49,160 lbs.
g. " ½ in. specimens from plates		48,000 lbs.	*Less than* 24 *in.*, 48,000 lbs. *Over* 24 *in.*....46,000 lbs.	8 *to* 24 *in. wide*....48,000 lbs. *Over* 24 *in. wide*....46,000 lbs.	
Elongation:		Per cent.	Per cent.	Per cent.	
h. Of *b*, full length		12.5	12.5	*For bars* ⅝ *in. thick or less*....15	
i. Of *d*, in 8 in		18	18	" *over* " 12	
k. Of *f*, "		15	15	*From* 8 *to* 24 *in*....12	
l. Of *g*, "		15	*Less than* 24 *in*....15. *Over* 24 *in*....10.	" 24 *to* 36 *in*....10 " 36 *to* 48 *in*.... 8 *Over* 48 *in*.... 5	

with the positive knowledge in advance that the mills offering to supply the material had never made any having the desired qualities. The bidders trusted to good luck to get through with their contracts as easily as possible.

Qualities.	Pittsburgh Mills.	Eastern Mills.
a. Elastic limit	Lbs. Over 27,500	Lbs. 24,000
Tensile Strength:		
b. Of full-sized bar under 4½ in. square	50,500 to 54,500	48,000
c. Of " " over " "	50,800	46,000
d. Of ½-in. specimen from *b*	53,000 to 56,000	49,000
e. Of " " " *c*	50,500 to 53,000	48,750
f. Of " " " angles and shapes	50,300	49,000
g. Of " " plates	49,700	48,000
Elongation:	Per cent.	Per cent.
h. Of *b* in full length	25 to 18	16
h. Of *c* " "	12	14
i. Of *d* in 8 inches	28 to 22	26
i. Of *e* in "	22	23.5
k. Of *f* in "	18	12
l. Of *g* in "	17	14

An engineer, undoubtedly, has the right and privilege to draw up specifications for wrought-iron and steel, the requirements of which can be fulfilled by certain mills only, and the fact that an engineer calls for bids in accordance with his specifications does not make it necessary for all works to bid. Such a view would be a positive wrong in some cases. The engineer has nothing to do with the broker or agent, and if the mills obtain their orders through such a person, they must take the responsibility of his action, and make sure in advance that he is competent and trustworthy.

Among drug-brokers and commission-merchants there is a well-understood code of ethics, and goods are invariably sold by specifications, samples and after inspection. When the goods furnished are not satisfactory, the lot is either rejected entirely, or the price is lowered to suit the degree of inferiority or damage. It would be to the advantage of all persons interested in the iron and steel business to grade the price of all material with reference to its qualities precisely as is done in other mercantile transactions.

As a rule, mill-men know before the inspector does whether any single piece is defective or not, and they should then immediately replace it before changing rolls, and not depend upon the leniency, good nature or ignorance of the inspector to have material accepted which they know to be defective. This has, unfortunately, been the method at some mills; and when an inspector does know his duties toward his engineer, he is criticized as incompetent, severe and unreasonable. Moreover, it is a common practice to try to force an engineer to accept a defective piece on the ground of the delay which would result from rejection. In our better mills, which make an honest effort to fulfill their contracts in a liberal manner, this case rarely arises. When material is inspected with proper promptness for surface-defects and correctness of shape, any shortcomings will be detected before changing rolls, and can be rectified instantly. Of course, when boys, without experience or authority to reject, are employed as inspectors, there never can be harmony between the mill and the engineer. What are styled "excessive rejections" never occur, unless the contractor is trying to keep as close to the minimum limits as possible, and then he has himself alone to blame, for it is self-evident that no contractor will allow an inspector or engineer to impose upon him or compel him to furnish material better than that contracted for. The fact that a mill has undertaken to furnish material better than it can make, does not permit the inspector to accept the material it can furnish, when both know it to be, though not "inferior material" in a general sense, yet below the requirements of the contract.

The drifting-test (page 681), is, as described by Mr. Hunt, "a practical test that shows the actual behavior of the metal, and can be made upon scrap-pieces without wasting merchantable stock and without expensive machine-shop work upon the test-specimens." Inasmuch as a rough sheared piece of scrap, with a hole punched in it, is used for this test, it will readily be seen that it can be made a correct test only when applied to very low steel, as other steels and all wrought-iron are considerably and injuriously affected by punching and shearing. Furthermore, the number of the blows of the sledge, its weight and height of fall or force of blow, are not mentioned or observed; neither is the duration of the test or the rapidity of repetition of blows,—all of which affect the results more or less. It is incorrect, therefore, to claim for this test that "it presents the further advantage that exact data can be obtained for records regarding it," and

the opinion that it "should be one of the standard tests of wrought-iron and steel plate and shapes" is not well founded.

Nowhere have any series of drifting-tests been made and published, except by Riley, of Scotland, which would allow us to assume that we are well informed about their practicability and the relation between the results obtained and the properties of the materials as determined by other tests. It is premature to advise the general and immediate adoption of the drifting-test.

For this reason it is not on the list of tests proposed for scientific testing in the "Report of Committee on Standard Tests and Methods of Testing Materials," *Trans.* A. S. Mech. Eng., xi., pp. 628 and 629.

Mr. Hunt says, also, that the hardening or quenching test, as proposed by the above committee, is of "doubtful utility, except when carefully made . . . . the amount of hardening depending not only upon the hardness in the steel, but also upon the temperature to which it is heated, and the character of the solution into which it is plunged to quench it." In describing the hardening-test, the report already cited says (p. 628):

"9. Hardening-tests are to be made by making the foregoing bending-tests, only carrying them on to rupture, and reading the angle at which rupture occurs; then other pieces, exactly similar to those thus tested, are to be heated carefully to a *fair red-heat*, and then plunged into water at a temperature of 70° to 75° F. [Erroneously printed "32° to 40° F.," but subsequently corrected by circular.] These quenched pieces are then to be tested precisely as those tested in their natural condition, and the angle through which they bend before fracture occurred is to be carefully noted; the difference in the amount of possible flexure in the two cases represents the amount of hardening produced."

It will be generally admitted that tests are worthless unless carefully made, and a "fair red-heat" and "water at 32° to 40° F." are probably sufficiently clear to most engineers who know how to test to prevent errors or misunderstanding on their part.

Mr. Hunt states (pp. 681, 682) that "the bending-test for soft structural steel is, by itself, not a crucial test of quality," and also that "it need hardly be remarked here that the nicked bending-test is not applicable to steel except under special conditions, as in rivets."

All authorities on steel are agreed as to the value of the bending and nicked bending-tests of steel, some even going so far as to assert that chemical analyses may be omitted when they are properly made, as by them the presence of undesirable elements in too great proportions can be determined. For proof of this, I refer to the report to

the Department of Railroads of Prussia, by Mr. George Mehrtens, Bromberg, May, 1891; to a lecture by Prof. R. Krohn before the Verein Deutscher Ingenieure, and published in their official organ, the *Zeitschrift*, in 1891; also to the "Report on the Use of Low Steels for Structural Purposes," made to and published by the Verein Oesterr. Ing. u. Architekten, 1891; also Lebasteur's *Sur l'Acier*, 1884, and many other authorities.

All of these reports demonstrate the great importance of bending-tests and point out their searching character, and particularly emphasize the value of the nicked bending-test for steel.

Mr. Hunt observes (page 683), that the "bulging-test recommended by Kirkaldy, though not often used in this country, is a useful, practical measure of the ductility of the metal, and especially adaptable for testing soft flange-steel."

Kirkaldy alone makes and recommends this peculiar test, and it is never made in this country. Hot flanging-tests are regularly made to determine flanging-qualities of boiler-steels, but it is not clear how a cold bulging-test can determine the qualities of the same material when hot. As there are no published results of buckling-tests, comparisons are out of the question; and as the relation between "bulging" and ductility has never been investigated or determined, the amount of bulging cannot be called a "measure" of ductility.

The results of a bulging-test depend upon the size of plate, of opening, of plunger, of edges of all three, upon thickness of plate, upon speed of test, and other matters, so that the effect of variation of each must be determined before any precision can be claimed for it. No such determinations have, as yet, been made.

Mr. Hunt's paper nowhere mentions one of the most important and crucial tests that can be made for the purpose of determining and measuring ductility, viz., the "drop-test." This and its value are most fully explained by Prof. A. Martens, in his recent publication, in the reports of the Royal Testing Laboratory, at Charlottenburg, Berlin, for 1891, in which he details experiments made during the past six years. This extended series proves that drop-tests, when properly carried out, can be made a measure of ductility and resilience, and are probably the readiest and quickest tests that can be applied.

Mr. Hunt observes (page 687), that "the only satisfactory method of testing finished structural open-hearth or Bessemer steel, must be based on the melts as the units of quantity." It has been demonstrated, time and again, that reheating and subsequent treatment may

produce greater differences in material than originally existed between the different melts or blows, and, for published proof of this, I again refer to Prof. Krohn's admirable paper on this subject, and particularly to that part of it relating to differences due to reheating and rolling.

With all the experience, published and personal, now available, there can be no doubt that each furnace-charge of blooms or billets should be tested to determine the effect of reheating and eliminate such material as has been injured. A simple test—especially either the bending- or nicked bending-test is the best for this case—should be made on each heat in a furnace-charge, and, furthermore, each largely varying shape should be again tested for further changes, precisely as is done in the case of wrought-iron.

The mill-manager can, however, reduce the number of tests to a minimum by charging all furnaces according to heat-numbers. Billet-tests do not, except in a few cases, represent the qualities of material after rolling; and although they are the best guide to the mill-manager as to the results he may expect from the steel after rolling, the engineer is hardly concerned with them, as all he cares to know is the quality of the material in the shape in which he uses it. Experience has proved that certain processes and methods are apt to produce non-uniformity of results and occasional failures, and hence the engineer must have the further privilege of saying whether he will accept material made by them; for if he were to call for tests, sufficient in number and kind, to eliminate the accidentally inferior material, he would delay the work considerably, and be considered and characterized as unnecessarily severe and unreasonable.

Notwithstanding all the claims of steel-makers, it is well known that different processes do not invariably assure equally uniform and reliable results. Hence, the engineer should have the privilege at least of knowing by which process the material he buys, has been made. This privilege does not, however, carry with it the right of the inspector to interfere in any manner with the metallurgical department, although the right to be present at all operations should be conceded, irrespective of whether he understands what is going on or not. It does not seem probable that any reputable works would urge objections to this, although they might, and with reason, object to the presence of a meddlesome person.

The above general remarks represent pretty closely the condition of affairs in continental Europe, where by far the larger proportion of output is used by the governments, and the method has been found to give universal satisfaction.

There is little doubt that similar methods will be introduced some day in this country as well, but only after engineers insist upon having their specifications followed as closely as is done in work for our government, without conceding point after point, as they are crowded by the contractors below them, and the board of managers or officers above them. It is generally confessed that our government always gets what is called for, and that there is no attempt to evade any number or kind of tests, although the specifications are more difficult to fill than those of any bridge or railroad engineer. This is because all the works know that objections against the laws and principles laid down are unavailing, and that departments are immovable.

Under section v., on page 691, of Mr. Hunt's paper, the term "elastic limit" is repeatedly used when "yield-point" is meant, and in one line it is stated that this factor "is not always a safe criterion of the quality of iron or steel," while further on failure to observe the same is declared to be "an unwise omission," etc. In Unwin's *Testing of Materials of Construction*, 1888, p. 6, we read: "The modulus of elasticity and limit of elasticity, once directly ascertained by experiment, would furnish all the data necessary to the engineer in applying the theory of elasticity and fixing the working-limits of stress." And again, on page 7: "For experience has fixed the limits of working-stress far within the primitive elastic limit;" and "if the proper working-stress is to be determined on rational principles, the study of the conditions in which the elastic limits change becomes essential." See, also, *Ueber die Veraenderung der Elasticitaetsgraenze und die Festigkeit des Eisens und Stahls*, by J. Bauschinger, Munich, 1886.

On page 691, Mr. Hunt says: "In steel the reduction of area is often the best indicator of ductility," but no explanation is given why this is not always the case. If it is not always the best indicator of ductility, it is a very uncertain and unsatisfactory result to observe, and, I think, might be neglected altogether. Moreover, it is very difficult to measure the exact section of the fracture, because it generally assumes, in the case of cylindrical and rectangular test-pieces, a shape or position not symmetrical, the surface of fracture being sometimes approximately a plane, sometimes oblique, sometimes concave on one piece and correspondingly convex on the other, sometimes partly a split as well as a break, etc. It is scarcely possible to make measurements in some of these cases, at least, on which important deductions could be safely based.

Furthermore, the reduction depends so much upon local condi-

tions and the external character of the test-piece, that the attempt to draw conclusions of any value in the case of structural material is worse than useless, since it is misleading. This is as true of steel as of wrought-iron, and no distinction should be made between them in this respect.

On page 692, the difference which apparently exists between duplicate tests of structural material is pointed out by referring to the tabulated results given in the report of the Committee of the American Society of Mechanical Engineers. But the results given in these tables were not obtained from structural material, but from different kinds specially prepared, most of which was cold-rolled Norway iron.

Such great differences are not found in good qualities of structural material, and, moreover, some of these differences are due to the fact that these tests had been made by many operators on all kinds of testing-machines.

On page 693, it is said that iron having a tenacity of 52,000 pounds per square inch "should be carefully tested for welding, as it is apt to be dry and incapable of welding strongly."

This statement is not corroborated by experience. Iron showing more than 25 per cent. elongation in eight inches is regularly made in the Lehigh Valley and at Pittsburgh, and is the best kind of material for welding, as can be proved by innumerable tests readily available. Nor do these irons of unusual ductility contain steel scrap; they are made of carefully-selected stock of muck-bar, containing, in some cases, as much as 30 per cent. of muck-bar made from Bessemer pig.

MR. HUNT: I agree entirely with Dr. Raymond's general statement that, "whenever the principle of a true, impartial inspection is introduced the value of the result rises," and that inspection, which is not separate from the manufacturing department, is neither conclusive nor satisfactory. Most engineers are very loath to put the decisive acceptance or rejection of material in the hands of parties pecuniarily interested in the result.

I am glad to see that Dr. Dudley, and several others who have discussed my paper, agree with my general proposition, that specifications should be drawn broad enough to include all good material, and only exclude that which the engineer considers not suitable for the purpose. As Dr. Raymond puts it, we should demand less and get what we demand, putting our severity in the right rather than in the wrong place, and making it the duty of the inspector

simply to determine whether the material answers the requirements of the specifications.

The question remains, what series of tests shall be made, and what results shall be required from them as the crucial conditions of the proposed new standard specifications?

Upon this question I hope for much light from the intelligent discussion which may be expected at the Congress of Engineers at the Columbian Fair next year. Certainly, there is a field here for work of very great public value.

Some of the individual points made in the discussion of my paper may be noticed here.

Of course, I do not deny that the Pottstown Iron Company, as well as other large structural iron and steel concerns, have "frequently made wrought-iron satisfactory to their customers, and that without inspection." I would, however, call attention to the fact that often the direct customers of the mills are not the users who eventually find out the satisfactory or unsatisfactory qualities of the material in actual work. It must be remembered, also, that the specifications and their various requirements are very rarely made up by the inspectors, but usually by the engineers who design the structure for which the material is required. With a specification drawn broad enough, and yet with intelligent precautions to exclude poor material, and permitting sufficient opportunities to prove the quality of the material up to the requirements of the specification (a condition of affairs practically impossible when only four tensile tests to the car-load are prescribed, irrespective of the heterogeneous quality of the material composing the lot), I agree with the position taken by the mill-people that the acceptance of material at the mills by the duly-accredited inspectors for the engineers, should be final. There are, however, several considerations which may modify this general statement. So far as my own experience goes, with the comparatively small amount of material which I have known to be rejected, subsequent to acceptance by the inspectors at the mills and shipment, the rejection has been due to one of the following causes:

1. The non-homogeneity of the steel, which has not been discovered by the few tests prescribed or permitted to the inspector to determine the quality of the material.

2. The failure of the test-requirements of the specification to develop the particular faults which the material has afterwards shown.

3. Surface-defects, which should manifestly condemn the material, but which have been either carelessly overlooked by the inspectors,

or were so concealed at the time of inspection by ice, snow, mud, or oxide of iron scale as to prevent their discovery until after subsequent handling had developed them. The most common form of this trouble is the hidden flaws in pin-steel, which are only shown in their exact character after the pins have been turned to size; or, seams due to the "piping" at the heads of steel ingots, which have not been cropped down low enough to exclude the entire "pipe," and which are developed in the upsetting of the steel in the heads of eye-bars, or in the punching or shearing of plates, etc.

4. Mal-formed material. This is especially a cause of trouble with plates of iron and steel, such as universal-mill plates warped in cooling, or with wide steel plates bent by improper handling while hot, or buckled by unskilful rolling.

The decision, whether such material should be accepted, is often a cause of considerable friction at the mill. The more rigid way of looking at the matter is, that in all these cases the material manifestly and clearly does not answer all the requirements of the specification. On the other hand, the answer of the mill-people, which is well known by the experienced inspector to be true, is, that it is impossible to get all the material absolutely uniform in quality, in surface, and in shape, not only with such large plates as quoted by my friend, Mr. Morris, "8 feet wide and 75 feet long," but also with the ordinary sizes and shapes of material turned out by all mills; and that, should the rejection of material include all such cases, the manufacturer would, necessarily and properly, in the long run, charge a much greater price for such severity in the specifications.

Mr. Morris says it is the custom of the Pottstown Iron Company, "for its own information," to make four chemical and five mechanical tests of each melt of steel, and I cheerfully testify to the care and value of these tests to which he refers. At the same time, this does not seem to me a reason why the customer should be satisfied with one or two tests per day, as he advises.

I believe the ideal way of testing steel would be the supervision by the inspectors of the tests referred to by Mr. Morris, and the further testing of crop-ends from the finished material by a standard method yet to be devised, which can be executed cheaply and readily enough to allow a large number of such tests to be made to prove the homogeneity of the metal.

Mr. Morris asks: "Why should the tests for steel be so much more rigid than for iron destined for the same purpose?" Some of the reasons are as follows: Experience shows that the acceptable

qualities of one melt of steel offer no absolute guarantee that the next melt to it, even though made of the same stock, will be equally satisfactory. Upon this point Mr. Morris has himself touched in his argument regarding the relative merits of the Bessemer and open-hearth steel.

Again, good wrought-iron, in plates and angles, has a narrow range (from 25,000 to 28,000 pounds) in elastic limit per square inch, and a tensile strength of from 46,000 to 52,000 pounds per square inch; whereas for steel, the range in elastic limit is from 27,000 pounds to 80,000 pounds, and in tensile strength from 48,000 to 120,000 pounds per square inch, with corresponding variations in ductility. Moreover, steel is much more susceptible than wrought-iron to widely varying effects of treatment, by hardening, cold-rolling or over-heating.

It is now almost universally recognized that soft steel, if properly made and of good quality, is for many purposes a satisfactory substitute for wrought-iron, being capable of standing the same shop-treatment as wrought-iron and of bearing safely loads of ten per cent. heavier unit-strains. But the conviction is equally general that poor steel, or an unsuitable grade of steel, is a very dangerous substitute for wrought-iron, even under the same unit-strains.

For this reason it is advisable to make more rigid requirements in selecting material which may range between the brittleness of glass and a ductility greater than that of wrought-iron.

I am already on record in the *Transactions* of the American Institute of Mining Engineers as not agreeing with Mr. Morris that the Bessemer is a better process for the manufacture of steel than the open-hearth. My experience for the last few years further confirms my opinion, and I find the literature of the subject to favor very strongly the superiority of open-hearth over Bessemer steel, as regards particularly its uniformity. The same difference has been recognized by most engineers in their specifications.

Nor can I agree with Mr. Morris that basic Bessemer or basic open-hearth steel should not require more careful testing than acid steel. Undoubtedly all will admit the undisputed danger of the high percentage of phosphorus in the stock used for the basic process, which has to be eliminated from the product; and to prove that it has been eliminated, a sufficient number of tests should certainly be made, beyond those required in the acid open-hearth process, where the stock used is low in phosphorus to start. Again, to eliminate a large amount of phosphorus it is necessary in the basic pro-

cess to run the carbon very low, and the fully purified basic metal, carrying phosphorus down to below .01 per cent., shows markedly decreased tensile strength. In order to answer requirements of 60,000 to 70,000 pounds tensile strength in this class of steel, it is necessary to raise the carbon to a point considerably above that in acid steel of the same tensile strength. This makes a metal much more subject to hardening by cold-rolling or other methods, and it requires more testing to prove the quality to be uniform.

I do not agree with Mr. Morris that "soft steel, either acid or basic, will weld as well as iron." While I admit that such material may be made to stick together with a great degree of tenacity, at the same time there is not the interlocking of the fibers of the metal due to the presence of slag, which is found in the true welding of wrought-iron. I have never seen so-called "welded" pieces of steel pulled apart in a testing machine or otherwise broken at the joint which have not shown a smooth cleavage-plane, as it were, such as in iron would be condemned as an imperfect weld. My experience in this matter leads me to agree with the position taken by Mr. William Metcalf in his paper upon Steel in the *Transactions* of the American Society of Civil Engineers, vol. xvi., page 301. Mr. Metcalf says, "I do not believe steel can be welded."

To the final comments of Mr. Morris, I would reply that inspectors will accept the mill-tests if they see them made. There are comparatively few men available as inspectors who have had a really valuable experience in the manufacture of iron. My own experience does not show the advantage of the "practical blacksmith" as an inspector; and I have had some little experience with men of this class among more than a hundred inspectors whom I have had, directly or indirectly, in my employ. Practical training in the mills is a valuable preparation, and one that certainly should be looked for in selecting inspectors of mill materials.

Standard dimensions for test-pieces are almost impracticable. If the further condition be made, that the test-pieces must be cut from the actual material to be furnished, it would be difficult to furnish a standard dimension for test pieces, for instance, equally applicable to 2-inch x 2-inch x ¼-inch angles and 8-inch x 2-inch eye-bars.

I agree with Mr. Henning as to the inadequacy of the Bridge Builders' Specifications in the number of tests called for, and in the requirements for these tests.

Mr. Henning has also pointed out the difference in the general

qualities of wrought-iron from the Pittsburgh district using Lake ores and coke-pig, and that from the Eastern mills using a larger portion of foreign ores and anthracite-pig. While there are one or two mills, notably that of the Catasauqua Iron Company, which form exceptions in the high tensile strength and other good qualities of their product, the general statement regarding the quality of metal made in the Pittsburgh district and the Eastern mills, has held true in the past. However, since the failure of natural gas in the puddling-furnaces, and the greater attention paid to the manufacture of steel over that of wrought-iron, the difference does not occur to such a marked degree to-day as in the past.

Mr. Henning has pointed out that two methods of testing are under discussion, namely "scientific testing," treated in the report of the Committee on Standard Tests and Methods of Testing Materials (*Transactions* of the American Society of Mechanical Engineers, vol. xi.), and the commercial methods of testing, which were the subject of my paper. I agree with him as to the value of the hardening and quench-tests as proposed by the Mechanical Engineers for "scientific testing." The disadvantages I referred to are connected with commercial testing in the mills, where the difficulties and anomalies I mentioned as affecting the quench-test are chiefly encountered.

My experience has further confirmed the statement made regarding the bending test, that it is not a crucial test; while as an additional test it is certainly a good one. For determining the difference between good and bad material this test alone is not in my judgment sufficient; as a considerable amount of material which gives in the tensile test a crystalline fracture and low percentage of reduction of area, will bend over on itself satisfactorily.

The drop test, for certain grades of material, such as steel axles, is certainly a suitable one, and should have been referred to in my paper as especially adapted for heavy sections.

I agree with Mr. Henning as to the advisability of a more careful method of testing material, by each re-heating furnace-charge as well as by each melt. This accords with the general statement made already that a further series of tests as to the homogeneity of the material, and its freedom from injury by treatment in hammering, rolling, etc., from the ingot to the finished material, is an important necessity.

A HAND-TELESCOPE FOR STADIA-WORK.

BY ROBERT H. RICHARDS, BOSTON, MASS.

(Glen Summit Meeting, October, 1891.)

If one holds up a prism, or wedge of glass, with narrow angle, say 1° to 2°, and compares the transmitted image with the image seen above or below the prism, the former will be found to be thrown to one side by an amount varying with the angle of the wedge. Speaking of the two rays as the direct ray and the bent ray, we may say, that when the bisecting plane of the prism is at right angles to the line of sight, the angle between the direct ray and the bent ray will be constant for any given prism.

If now we place a prism or wedge of glass in such a position that it half-covers the objective of a telescope, we shall obtain on looking through it two images of every object seen—one image by the direct ray, which comes through the uncovered half of the objective, the other which comes through the prism, and is the image by the bent ray. The angle of divergence of these two rays will be constant and unalterable, whether the telescope is directed to a near object, with its eye-piece at increased distance from its objective, or upon a distant object, with eye-piece nearer to the objective. That is to say, if the "throw" or apparent dislocation of the image is one foot in one hundred feet it will be two feet in two hundred, ten feet in a thousand, and so on.

The usual form of stadia-telescope has at the focal point of the objective two spider-lines, placed at a definite distance apart and intended, let us say, to represent a throw of one foot in one hundred feet. But as the distance between lines remains the same, while the distance from the objective to the webs differs with every variation in the distance of observed objects, it follows that the angle between the lines of sight which these two spider-lines define cannot be constant, but must vary with every increase or decrease of distance between the instrument and the object viewed.

If in Fig. 1 we represent three positions of the spider-lines by W_1, W_2, W_3, and three objects corresponding to those positions by O_1, O_2, O_3, at near, medium and distant positions, we see at

a glance from the figure that there is no constant angle represented by the spider-lines, and that the only way to graduate a rod for the practice of stadia-measurement by this method is to determine values for one foot at a sufficient number of distances, and to provide the rod with a graded scale accordingly.

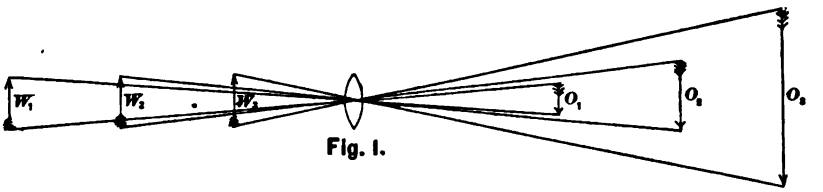


Fig. 1.

The prismatic stadia-telescope, on the other hand, has a constant angle for all distances and all focal lengths; and when the factor has once been obtained, it may be used to graduate a rod with uniform scale from end to end.

Again, the usual stadia-instrument involves two points of observation. The operator adjusts the lower spider-line on the zero of the rod and then observes the reading of the upper line on the rod. There are, therefore, two personal equations in the operation of taking readings. With the prismatic stadia-telescope, on the other hand, both observations are made at once, just as the sailor, in taking the altitude of the sun at sea, brings the sun's image to the horizon and observes the contact. Only one personal equation is thus involved.

The spider-line stadia-telescope cannot be used by simply holding it in the hand, but requires a firm support; for, if it were used in the hand, the first line would wander from the zero on the rod before the reading of the second line had been taken. The prismatic stadia-telescope, on the contrary, can be used in the hand, just as a sextant is. Nevertheless, as the readings to be made with it are much finer than those of the sextant, a support will be preferred for most purposes.

The one great advantage of the spider-line over the prismatic telescope is, that it uses the full light of the whole objective at all times, while the prismatic has only half-light. This objection to the latter is completely removed when the spacing target-rod is used; and it is fairly well met by the employment of self-reading rods, to which reference will be made below.

In adapting the prismatic stadia-telescope to the needs of the surveyor, several important matters have to be determined, relating to the prism and also to the telescope and the rod.

In my experiments I have combined a telescope of 30 diameters magnifying-power with a prism of one foot throw to 100 feet, and also with a prism of one foot throw to 150 feet. I shall speak of these two combinations respectively as 30 d : 100 and 30 d : 150. I have also combined a telescope of 20 diameters magnifying-power with three prisms, throwing respectively 1 foot in 50 feet; 1 foot in 100 feet; and 1 foot in 150 feet; and I shall speak of these combinations as 20 d : 50; 20 d : 100; and 20 d : 150. Finally, I have combined a telescope of 10 diameters magnifying-power with prisms of 1 foot to 50 and 1 foot to 100; and I shall call these combinations 10 d : 50 and 10 d : 100.

The first fact I encountered in these experiments was, that 10 d : 50 and 10 d : 100 can be used with uncorrected prisms, since the amount of color does not seriously injure the observation; while no prism not thoroughly achromatic was found satisfactory for either the 20 d or the 30 d telescope.

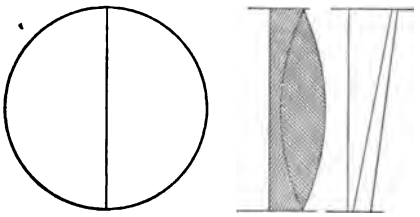


Fig. 2.

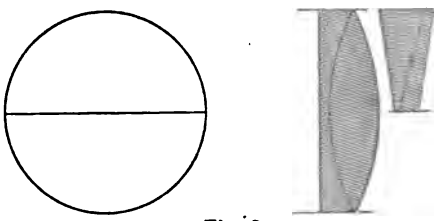


Fig. 3.

Fig. 2 represents what appears to me to be the rational and proper mode of combining the prism and the objective; while Fig. 3 shows an improper combination. In Fig. 2 the bent rays are exact counterparts of the direct rays, while in the adjustment of Fig. 3 this is not the case.

Choice of a Prism.—For hand-use a wide prism is preferred, say 1 foot displacement in 50 feet distance. For a fixed telescope 1 : 100 or 1 : 150 would be preferred, according to the distances to be sighted and the length of the rod to be used. A prism throwing 1 : 150 will permit readings at 1500 feet on an 11-foot rod, while a 1 : 100 prism will read only to 1000 feet on the same rod. On the other hand, where a standard steel tape with sliding targets is used instead of a rod, we may say that the wider-angle prism will give more accurate work than the narrower.

Choice of a Telescope.—Since a large field is not needed, we have to deal with two considerations only: the lightness of the instru-

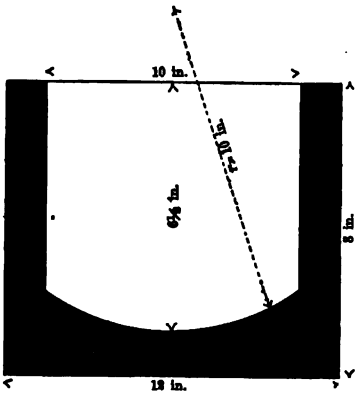


Fig. 4.

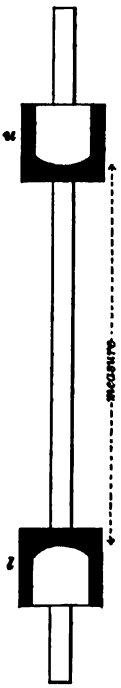


Fig. 5.

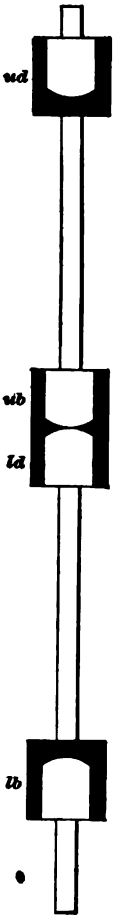


Fig. 6.

ment and its power. If the former consideration governs the choice, I would recommend a light 10 *d* or a light 20 *d* telescope; if the latter, I would recommend a 20 *d* or a 30 *d* telescope.

Choice of Rods.—In this direction I believe I have settled the question as regards the choice of a spacing-rod with sliding targets; but I do not feel so sure with respect to the self-reading rods.

The spacing-rod represented in Figs. 4 to 9 is so pre-eminently superior to every other which I have tried that I strongly recommend it.

This target stands out strong and bright under any circumstances. It can be read against a dark background in the woods and also against a light sky background on a hilltop.

In Fig. 5 the rod is represented as seen by the eye. Fig. 6 shows it as seen by the prismatic stadia-telescope. In this figure *ud ld* are the upper and lower images by the direct ray, and *ub*, *lb* are the upper and lower images by the bent ray. The observer of course neglects *ud* and *lb* and uses for his contact *ub* and *ld*. The important advantages of this form of contact are shown in the subsequent figures.

Fig. 7 represents the targets approaching contact; Fig. 9 shows the images lapping, in which case there is a bright lens-shaped image, indicating the amount of lap. Upon its appearance the assistant is signalled to draw the targets slowly apart; and the instant the white lens becomes

invisible the reading, Fig. 8, is taken. The disappearance of this white lens gives a close and positive reading. The results stated below were obtained with this reading.

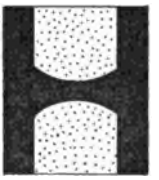


Fig. 7.

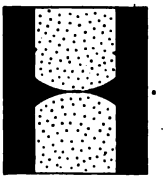


Fig. 8.

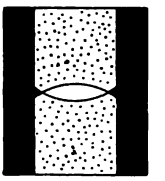


Fig. 9.

A target of the size shown in Fig. 4 can be read easily at 2000 feet distance with a 20 *d* telescope, in an even atmosphere. A much smaller target will suffice for short sights. The design of Fig. 4 has more strong points in its favor than any other yet tried by me.

The spacing-targets, Figs. 4 to 9, may be used on a steel tape if the following conditions are observed: The tape must be held at right angles to the line of sight. If the right-angle is at one of the targets, it will be found easier to reproduce than at a point half-way between the targets. For distances much greater than 2000 feet a larger target or a higher-power telescope, or both, will be needed.

Fig. 10 represents a tape, as used by me on a 1970-foot sight with the above targets and a 20 *d* telescope.

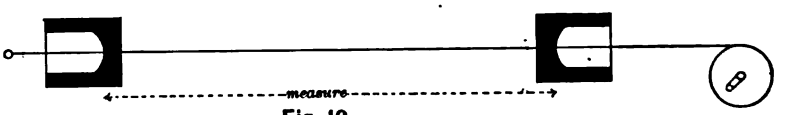
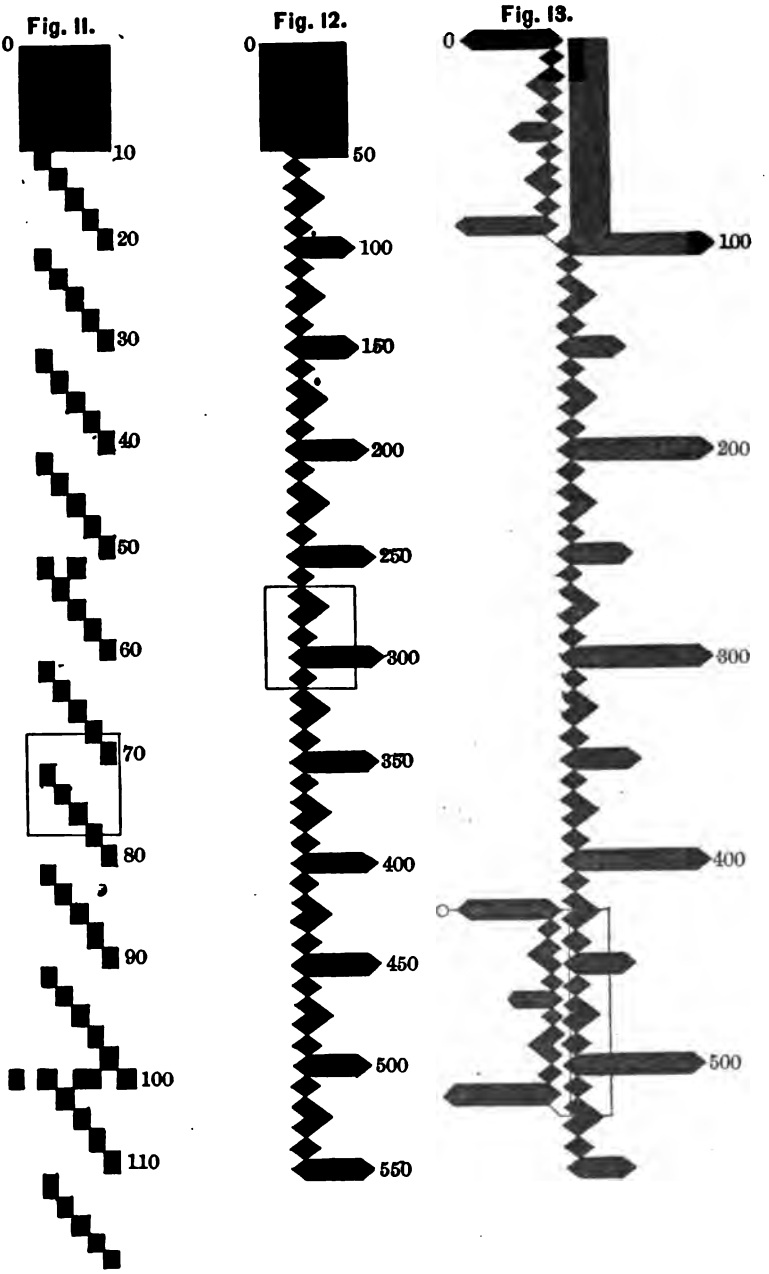


Fig. 10.

Self-reading Targets.—Three forms which have been experimented upon are represented in Figs. 11, 12 and 13.

Fig. 11 gives good results with a 10 *d*: 50 combination for short distances. It may be graduated, as shown, with 2-foot unit or with 5-foot or 10-foot unit. In the latter case, it would be used for long distances with a 20 *d*: 100 combination up to 30 *d*: 150 combination; but the individual feet would have to be divided by the eye. The skeleton second image on it reads 68 feet.

Fig. 12 may be graduated with 1-foot, $2_{\frac{1}{2}}$foot, 5-foot, or 10-foot units, according as a short-distance or a long-distance rod is wanted. The skeleton-reading of the second image upon it indicates 265 feet.



Possibly, for certain distances, this graduation may be found easier to read than that of Fig. 11.

Fig. 13 represents a target-rod with an optical vernier. The space from 0 to 100 feet on the rod is graduated into 11 parts, while the spaces from 100 to 200, also from 200 to 300, and so on down, are graduated into 10 parts. Moreover, the space from 0 to 100, which we may call the vernier, is graduated on the opposite side to the other readings, so that the second image can bring it down and give a vernier-reading between the two images. The partial skeleton second image gives a reading of 423 feet, no dividing by the eye being required. By this system an 11-foot rod may be made for a $30\,d : 100$ combination, which will actually give readings of individual feet at 1000 feet distance.

The vernier-reading is the most fascinating idea I have met in my investigations. Whether it is really practical, can only be decided in the field. The images which are important for the reading are half-light images, and therefore dim, while with the other two forms of target a full-light reading is obtained.

Limits of Error.—Partly by reason of the limited time at my command for experiments, and partly because, every time I went out to get definite records of practice, I made some discovery which led to an improvement in the apparatus, I can only promise at this time to give detailed figures at an early day as a supplement to this paper. The figures I am now prepared to publish were taken with a curved spacing-target of the design shown in Figs. 5 to 9.

With a $20\,d : 150$ combination, holding the rod at 100 feet distance, I obtained five separate readings of the distance in feet between the targets, which were 0.653; 0.654; 0.653; 0.653; 0.653. With a $30\,d : 150$ combination I made these four successive readings at 100 feet distance: 0.675; 0.675; 0.675; 0.675; all with the spacing-target described in Figs. 4 to 9.

If I had made an error of 0.001 foot upon the rod, this would correspond to an error of 0.15 per cent. or 0.15 foot in 100 feet. But I did not make this error in four readings with $30\,d : 150$, and only made it once in five successive readings by the $20\,d : 150$ combination. If the prism had been a 1 : 100 prism, the error referred to would have been 0.1 foot in 100. If the prism had been a 1 : 50, the error would have been .05 foot in 100 feet. I cannot imagine a reason why the percentage-error at 1000 feet or 2000 feet, when the atmosphere is steady, should be any greater than it is at 100 feet. From the above considerations, I believe I am safe in saying that the error of the prismatic stadia is well inside of 0.1 per cent.

I hope to present, in the near future, some figures actually obtained for both short and long distances.

If one desires to determine the distance to a point without the trouble of sending an assistant there, it may be done with a pair of Wollaston camera-lucidas, as shown in Fig. 14, in which W_1 W_2 are the two cameras.

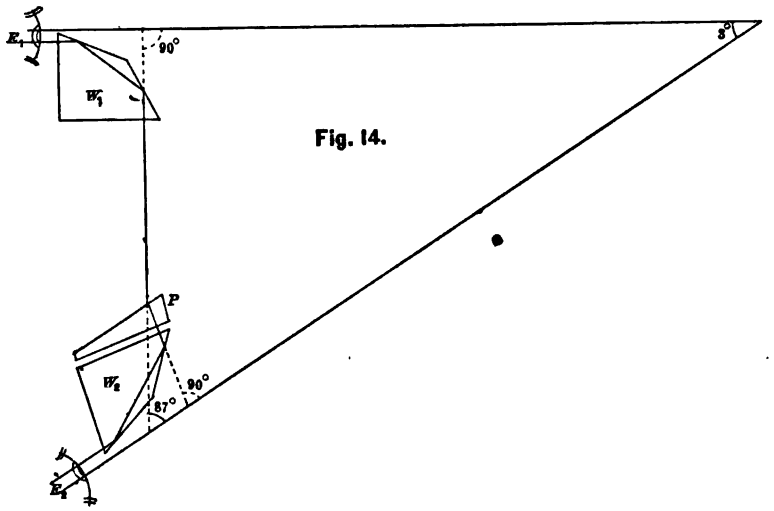


Fig. 14.

If the Wollastons both give 90°, then the requisite deviation from 90° may be obtained by inserting a prism, P, as shown. If, however, the two Wollaston angles add up to say 177° to 178°, then the apex-angle will be 2° or 3°, and will answer without the addition of another prism. After the two observers, E_1 E_2, have found their places, so that their respective images coincide, then they can measure the base-line between them by a prismatic stadia-telescope; and knowing the factor of the Wollastons, 3 feet or 4 feet to the hundred, the distance to the unknown point may be determined approximately by multiplying the observed distance between E_1 and E_2 by the factor,—3 feet to 100; 30 feet to 1000; 300 feet to 10,000; and so on.

In making my designs and in testing my instruments, I have been helped materially by Captain A. H. Russell and Messrs. J. Hays Gardiner, W. H. Weston, Franklin Knight, Luis T. Verges, and W. S. Hutchinson, friends to whom I wish to make acknowledgment.

PHOTOGRAPHIC AND CO-ORDINATE SURVEYING.

BY HENRY M. STANLEY, NOGALES, ARIZONA.

(Glen Summit Meeting, October, 1891.)

THE methods about to be described have been tested in practice, and are believed to be specially adapted to such extended topographical surveys as mining engineers in particular are sometimes called upon to execute. The assistance of Mr. H. O. Flipper in the preparation of this paper, and his practical authorship of the second part, is hereby acknowledged.

I. PHOTOGRAPHIC SURVEYING.

The writer was recently employed, together with Mr. H. O. Flipper, to survey the properties of the Plata-Reina de Sonora Mining and Milling Company, situated in the mining district of Las Planchas, District of Magdalena, State of Sonora, Mexico.

These properties, now owned by an American corporation, of which M. W. Evans, of New Orleans, is president, and Richard A. Pomeroy is resident superintendent, are famous in the history of Mexican mining, as the place where *planchas*, or *bolas de plata* were found about the beginning of the last century. Ward, the English *Chargé d'Affaires* in Mexico from 1826 to 1828, writing at that epoch, says:

"The idea [that the ores in the north of Mexico were richer than those in the south] probably originated in the discovery of the famous *bolas de plata* [balls of silver] of Arizona [Planchas de Plata de Arizona, a stock-ranch in northern Mexico], in the beginning of the last century, which was, and probably still is, believed in Europe to be one of those fables with which mining countries always abound. But the attention of the present government of Mexico having been drawn to the subject, a search was made in the viceregal archives, by order of the president, for the correspondence which was known to have taken place respecting it in the year 1736. This correspondence I have seen, and I have in my possession a certified copy of a decree of Philip the Fifth, dated Aranjuez, 28th May, 1741, the object of which was to terminate a prosecution, instituted by the Royal Fiscal, against the discoverers of Arizona, for having defrauded the treasury of the duties payable upon the masses of pure silver found there. The decree states the weight of the balls, sheets and other pieces of silver discovered (*bolas, planchas, y otras piezas de plata*), to have amounted to 165 arrobas, 8 pounds, in all, 4133 pounds; and mentions particularly one mass of pure silver, which weighed 108 arrobas (2700

pounds); and another of eleven arrobas, upon which duties had been actually paid by a Don Domingo Asmendi, which, as a great natural curiosity (*como cosa especial*) the king states ought to have been sent to Madrid. The decree ends by declaring the district of Arizona to be royal property, as a *criadero de plata* (a place in which, by some natural process, silver was created), an idea, to which the flexibility of the metal, when first extracted, seemed, in those times, to give some color of probability, and by directing it to be worked upon the royal account. This put a stop to the enterprise of individuals; the district was deserted; an attempt to send a sort of colony there failed; and in a few years the very name of Arizona was forgotten."

These properties are six in number, each 800 meters long and from 200 to 300 meters wide. The tract surveyed comprises about six square miles and is broken, mountainous country. It was desired that the work be done as quickly and as accurately as possible. Under these circumstances it was decided to triangulate the tract and use photography for the remainder of the work. The purpose of this paper is to describe that part of the work.

Photographing.—An 8 x 10 Eastman camera was used. Two cylindrical levels at right angles were placed on the bed, a single cylindrical level on the front board and another on the rear box. These two levels were placed parallel to the optical axis. In operation, the bed was carefully levelled, and the front and rear were made level by the vertical swing. This being done, the principal focal distance of the lens was carefully determined, the use of which will be explained further on. The geometrical center of the ground glass was then found, and the optical axis was made to pass through that center, when the apparatus was level. This, the normal position of the lens, was marked, and a scale was pasted on the front board, with its zero at that normal position. Horizontal and vertical lines were drawn through the center of the ground glass and means were provided for reproducing these lines on the negative. In the survey in question, celluloid films were used and notches were made in the carriers. These notches are photographed on the negative and lines joining them are the reproductions of the horizontal and vertical lines on the ground glass.

To take the picture, the camera is set up over the point and levelled and the landscape is carefully focused on the ground glass. If necessary, the front board may be raised or lowered, in order to get more or less of the landscape in the picture, in which case the amount by which the front board has been raised or lowered, as indicated by the scale, must be recorded in the notes. The height of the optical axis above the ground must also be measured and recorded. The following form of record was used:

STATION..... 18

FOCAL DISTANCE.....

View No.	Index.		Co-ordinates.		Height of Camera.	REMARKS.
	Hor.	Vert.	Hor.	Vert.		

The first picture was taken to the north and to the right of the meridian. The others were taken around the circle, one after the other, so as to lap slightly, care being taken to note the height of the instrument and the index-number for each picture.

Plotting.—This is illustrated in Fig. 1. The points marked "Corruscos," "Mejia," "N. Coneccion" and "N. Guadalupe" are points established by triangulation, at each of which pictures were taken. This sketch was originally constructed on a scale of one-half inch to one hundred and fifty feet. The focal distance of the lens used is 20.3 half-inches.

Desiring to locate the point "W. Ladera," we look over our pictures and find that this point is on VIII Mejia and I North Guadalupe. The first operation is to orient these pictures, *i.e.*, find and plot the position of the optical axis and of the picture in each case. On VIII Mejia, the point "Corruscos" is known. Mark it (1) on the back of the negative. With the dividers measure carefully the horizontal distance from this point to the vertical axis. From a diagonal scale of equal parts constructed on a half-inch, the horizontal distance is found to be + 5.53 half-inches, or units, and as it is to the right of the vertical axis it is marked plus. This distance is the altitude of a right-angled triangle whose base is the focal distance of the lens. The hypotheneuse is the distance from the optical center of the lens to the image of the known point on the picture, in this case, "Corruscos." Now as the tangent of the angle at the base of a right-angled triangle is equal to the altitude divided by the base, we have

$$\text{tangent of angle} = \frac{+ 5.53}{20.3} = + 0.27241.$$

Orientation angle = + 15° 15′.

This result is placed in the proper column of a table ruled as follows:

Photographic Survey of.....

SCALE, ½ inch = 150 ft. FOCAL DISTANCE, 20.3.

STATION.	View No.	Point No.	CO-ORDINATES.		Orientation Angle.	Location Angle.	Vertical Angle.	Height of Point above or below Station.	Height of Stat'n above or below Datum.	Height of Point above or below Datum.	Distance.	REMARKS.
			Horizontal.	Vertical.								
Mejia.....	VIII.	1	+ 5.53	.....	+ 15° 15′	.....	.....	.....	.....	.....	.....	Oriented on Corrusco Flag.
N. Guadalupe...	I.	1	+ 5.47	.....	+ 15° 04′	.....	.....	.....	.....	.....	.....	" " N. Coneccion "
Mejia	VIII.	2	— 3.80	+ 0.81	.....	— 10° 36′	+ 2° 15′	+ 222′.8	+ 500′	+ 722′.8	5668′	Location of W. Ladera Flag.
N. Guadalupe...	I.	2	+ 2.35	— 0.51	.....	+ 6° 36′	— 1° 26′	— 157′.3	+ 880′.1	+ 722′.8	6292′	" " " "

FIG. 1.—Sketch Illustrating Method of Plotting from Photographic Survey.

To plot the position of the optical axis, draw a line through the station Mejia making with the hypotheneuse, Mejia-Corruscos, an angle equal to — 15° 15′, *i.e.*, to the left. Make this line equal to 20.3 half-inches or units and it will be the optical axis. A line perpendicular to it at its extremity will be the position of the picture VIII Mejia. In the same manner the picture I N. Guadalupe may be oriented as shown on the sketch, using for the purpose point (1) of that picture, which is the flag at N. Coneccion. The calculation in this case is:

$$\text{Tangent of the angle} = \frac{+5.47}{20.3} = +0.26945.$$

Orientation-angle = + 15° 04′.

Having oriented the two pictures, we are now prepared to determine the positions of all points common to them both. On picture VIII Mejia we find that the point W. Ladera, point (2), is — 3.80 units to the left of the vertical axis and + 0.81 units above the horizontal axis. The same point (2) on the picture I N. Guadalupe is + 2.35 units to the right of the vertical axis and — 0.51 units below the horizon. These results are recorded on the calculation-sheet. Now

$$\frac{-3.80}{20.3} = \text{tangent of angle} = -0.18719.$$

Location-angle = — 10° 36′.

At the point Mejia draw the line Mejia-W. Ladera, making with the optical axis an angle equal to — 10° 36′, that is, to the left of the optical axis.

From picture I N. Guadalupe we have

$$\frac{+2.35}{20.3} = \text{tangent of angle} = +0.11576.$$

Location-angle = + 6° 36′.

At the point N. Guadalupe, draw the line N. Guadalupe—W. Ladera, making an angle with the optical axis of picture I N. Guadalupe equal to + 6° 36′, that is to the right of the optical axis. The intersection of the two lines Mejia-W. Ladera and N. Guadalupe-W. Ladera is the position of the point W. Ladera, which was required.

This point could have been located graphically. On the line

LAB lay off AB = — 3.80. Through B and Mejia draw the line Mejia-W. Ladera. On the line DFN lay off FE = + 2.35 and draw N. Guadalupe-E-W. Ladera. These two lines will intersect at W. Ladera. The distance Mejia-W. Ladera and also N. Guadalupe-W. Ladera are scaled from the map and placed in the proper column of the calculation-sheet.

To find the elevation of W. Ladera, or its depression, above or below Mejia and N. Guadalupe, we proceed as follows: Through the point B, draw BC perpendicular to B-Mejia and make it equal to + 0.81 units. Through Mejia and C, draw the line Mejia-C-M. At W. Ladera draw W. Ladera-M perpendicular to Mejia-W. Ladera. Then the line W. Ladera-M is the elevation of W. Ladera above Mejia, and it may be scaled and placed on the calculation-sheet.

At E draw DE perpendicular to the line N. Guadalupe-W. Ladera and equal to — 0.51 units, and at W. Ladera draw W. Ladera-G perpendicular to the same line. Through N. Guadalupe and D draw N. Guadalupe-D-G. Then W. Ladera-G is the depression of W. Ladera below N. Guadalupe. Scale this, and place it on the calculation-sheet.

To calculate the elevation:

In the right-angled triangle Mejia-A-B, the side

$$\text{Mejia-B} = \frac{20.3}{\cos 10^\circ\ 36'} = 20.6 \text{ units.}$$

In the right-angled triangle Mejia-B-C, the tangent of angle at base $= \frac{0.81}{20.6} = 0.03932$; the angle of elevation $= 2^\circ\ 15'$.

In the triangle Mejia-W. Ladera-M, the side W. Ladera-M = $5668 \times \tan 2^\circ\ 15' = 222.8$ feet.

In the triangle N. Guadalupe-F-E, the side N. Guadalupe-E = $\frac{20.3}{\cos 6^\circ\ 36'} = 20.4$ units.

In the triangle N. Guadalupe-D-E, the tangent of angle at base = $\frac{-0.51}{20.4} = -0.02500$; the angle of depression $= -1^\circ\ 26'$.

In the triangle N. Guadalupe-W. Ladera-G, the side W. Ladera-G $= 6292 \times \tan 1^\circ\ 26' = 157.3$ feet.

Therefore W. Ladera is 222.8 feet above Mejia and 157.3 feet below N. Guadalupe. If Mejia is 500 feet above the datum-plane, then W. Ladera is 722.8 feet and N. Guadalupe 880.1 feet above

the same plane. In the same manner any other point on these two photographs may be located graphically or by calculation.

The point W. Ladera may be determined from one photograph thus:

The side Corruscos-Mejia is known from the triangulation, and the bearing of Corruscos-W. Ladera from the field-notes. Then in the triangle Corruscos-Mejia-W. Ladera, we have the angle at Corruscos = 139° 45′. From the photograph VIII Mejia, we can find the angles 15° 15′ and 10° 36′ as before. Their sum 25° 51′ is the angle at Mejia. The angle at W. Ladera is equal to 180° — (139° 45′ + 25° 51′) = 14° 24′. The triangle can now be solved by the ordinary sine-formula, and the distances from Mejia and N. Guadalupe to W. Ladera and their bearings can be determined.

General Remarks.—The elements for determining the elevations or depressions of points above or below the plane of the optical axis may be measured directly in the field without taking the picture at all. For this purpose, the ground glass is ruled in squares, each side being equal to one unit of the scale, *i.e.*, to one-half inch, or one centimeter, etc. The focal distance of the lens must be expressed in the same unit, as so many half-inches, centimeters, etc. By shifting the front board, any point may be brought on the intersection of any horizontal and vertical lines, and its distances from the axes read in whole numbers. The index-readings being carefully noted for each point will enable the surveyor to reduce these elements to the same horizontal plane, or, which is the same thing, to the same height of instrument.

The entire survey may be made by the photographic method, distances, elevations and vertical and horizontal angles being determined as accurately and far more expeditiously than by any other method. For taking topography this method is invaluable. The photographs are superior to any sketch the ordinary surveyor can make, and contain all the necessary data for determining the contours of the landscape in the picture.

Any ordinary camera, modified as indicated, may be used, though it is believed that a 5 by 7 or 5 by 8 picture is as small as will give good results. As to lenses, one with a long focus is recommended. Celluloid films with a matt surface on the back are to be preferred to glass for obvious reasons. They are light and non-shrinkable, and the matt surface can be written upon with pen or pencil. While the plotting may be done directly from the negatives, it is at times difficult to recognize points on them. Paper positives shrink and

cannot be relied upon. To avoid these inconveniences, we tried positives on the celluloid films and they gave excellent results. If it is desired to use the negatives for other purposes, positives may be made on films and the measurements taken from these latter. Or a film may be thorougly cleansed and ruled as desired for a permanent scale.

As surveyors are not usually photographers, and as it is not always possible to obtain the services of an expert, the following notes from actual experience may be of some value. We have found that in order to obtain the best results at elevated stations, it is necessary to have the sun as low as possible and still be able to shield the lens from direct illumination. Our best negatives were made with the camera directed due west and so late in the afternoon that it was necessary to hold the slide of the plate-holder like a hood over the lens to protect it from the direct rays of the sun. Negatives taken with the sun high are unsatisfactory, as they lack contrast, unless it is cloudy, and even then the landscape is too evenly illuminated for the best results.

For development, any standard formula may be used, but the inconvenience of transporting balances, graduates, etc., or chemicals in solution, led us to adopt the following method, in which all materials are carried dry and the necessity of weighing and measuring is obviated. For developing 200 8 by 10 films, the following is ample:

1 ounce Hydroquinon.
1 ounce Eikonogen.
1 pound Sodium sulphite, crystals.
1 pound Potassium carbonate, dry.
3 ounces Chrome Alum.
1 ounce Citric acid.
5 pounds Sodium hyposulphite.

To prepare the baths for development, dissolve in about a pint of water 4 or 5 pieces of chrome alum, each about the size of a hazel-nut, and 1 or 2 pieces of citric acid of the size of a bean. The hypo bath is prepared as usual. When ready to develop, put into 5 or 6 ounces of water 20 to 30 drops of the saturated sulphite solution and 4 or 5 of the saturated carbonate solution. To this bath add of the two reducers as much as can be heaped on a 3-cent piece. If the picture does not soon appear, add another 5 drops of the carbonate solution. With this method chemical fog is impossible and over-exposure is under good control, while under-exposure can be remedied by soaking the film in the developer before adding the re-

ducers. Very good results were obtained from equal parts of Eiko and Hydro. As the Eiko gives energy and the Hydro density, the operator may modify the amount of each used to suit any particular case. The Eiko should be rubbed down as fine as possible in a mortar as it assists solution. After development, an immersion in the bath of chrome alum and citric acid will prevent stripping and help to clear the film.

II. Co-ordinate Surveying.

The preparation of this part of the present paper was suggested to Mr. H. O. Flipper by the article on the same subject by Mr. Henry F. Walling, in Van Nostrand's *Science Series* No. 72. Mr. Walling's article contains more of suggestion and recommendation than explanation of the system. Mr. Flipper has used extensively a system of co-ordinate surveying which seemed to fill, better than any other, the conditions of the work in which he was engaged (public land surveying in Mexico, under concessions), viz., expedition and cheapness, with an acceptable degree of accuracy; and he has placed at my disposal the following account of his method, for which I desire that full credit shall be given to him, professionally.

The first step is to make a reconnaissance of the section to be surveyed, select triangulation-points and erect at each a suitable signal. A base-line is then measured in a suitable place. (During the progress of the work, or at its termination, one or more additional base-lines are measured, to check the calculations. For these measurements, a 20-meter steel tape is used). The base is corrected for grade, pull, sag and temperature, and reduced to sea-level; and its astronomical bearing, or, better, its azimuth, is determined. From each triangulation-point the magnetic bearing (or preferably the azimuth, if the instrument is graduated from 0° to 360°) of every other flag that is visible is then observed. As only the angles of the triangles are wanted, it is not necessary to worry about local attraction or astronomical bearings or variation of the needle. Due care is taken to have all observations as correct as possible, and at least three observations are made for each determination. The following is an extract from actual notes, showing the form of record employed.

At the top of the page the date and the name of the point where the observations are made are noted. If the point is a triangulation-point, a small triangle is placed after it as in the instance given above. In the left-hand column are put the names of the points observed and after them the readings of the instrument, that is, the bearings or

EL MORO, Δ, JULY 12TH, 1890.

Points observed.	Direct Position.		Reverse Position.	
	Ver. I.	Ver. II.	Ver. I.	Ver. II.
San Juan	48° 10′ 30″	10′ 00″	48° 10′ 30″	10′ 00″
	10′ 00″	10′ 30″	10′ 30″	10′ 00″
	10′ 30″	10′ 30″	10′ 00″	10′ 30″
Mean	48° 10′ 20″	10′ 20″	48° 10′ 20″	10′ 10″
San Pedro	73° 14′ 30″	14′ 15″	73° 14′ 15″	14′ 30″
	14′ 15″	14′ 30″	14′ 15″	14′ 15″
	14′ 15″	14′ 30″	14′ 15″	14′ 30″
Mean	73° 14′ 20″	14′ 25″	73° 14′ 15″	14′ 25″

azimuths of those points, as seen from the point of observation. Similar columns may be provided for vertical angles when desired. The angles may be measured directly of course, and as often as desired; and it is evident that the angles themselves are not affected by local attraction of the magnetic needle.

With the base-line and the angles deduced from the bearings or azimuths, all the distances covered by the triangulation may be calculated. Every line may be calculated from two or more triangles, affording in this way a check on each by comparison of results. Each triangle may be corrected for spherical excess and the forward- and back-azimuth of every line computed. When the sections surveyed are comparatively small, and unless it is intended to combine a number of these into a large general map, as the counties of a State, for instance, it is not necessary to correct for spherical excess.

The triangles are calculated by the ordinary sine-formulas. The angles are computed from the azimuths by subtracting the first from the second, adding 360° to the latter if 0° intervenes. Instead of calculating the mean of each set of observations, calculate the mean of the repetitions, the angle corresponding to each position of the telescope and vernier, and take the mean of these as the result required. To illustrate, find the angle at El Moro between San Juan and San Pedro. We have

73° 14′ 20″	73° 14′ 25″	73° 14′ 15″	73° 14′ 25″	25° 04′ 00″
48 10 20	48 10 20	48 10 20	48 10 10	25 04 05
				25 03 55
25 04 00	25 04 05	25 03 55	25 04 15 and	25 04 15
				25 04 04

which is the angle at El Moro between San Juan and San Pedro.

Azimuths are here estimated toward the right from 0° at the north pole to 360° or 0° again. They may be estimated from the same point toward the left, or from 0° at the south point either right or left.

The calculations should be made in the following order: 1. Reduction of base-line; 2. Triangles of triangulation; 3. Azimuths of sides of triangles; 4. Co-ordinates of triangulation points; 5. Three-point problems; 6. Unimportant points by intersection; 7. Plotting; 8. Areas.

Printed blanks of the following form are used for calculating plane triangles, with seven-figure logarithms. The algebraic signs placed before the logarithms on this blank and others given in this paper indicate the operations to be performed, while those placed after them are the signs of the quantities themselves.

TRIANGLES.

A	Prieto		B	Rajon		C	Yaqui	
A C	Prieto-Yaqui			4.3547215				
B	53° 45′ 35″	sin −		9.9066286				
				4.4480929				4.4480929
C	44° 21′ 12″	sin +		9.8445276	A	81° 53′ 13″	sin +	9.9956315
				4.2926205				4.4437244
A B	Prieto-Rajon			19616.4 m	B C	Rajon-Yaqui		27779.5 m

A		B		C	
A C					
B	sin −				
C	sin +		A	sin +	
		m			m
A B			B C		

Spherical excess is computed by Legendre's formula,

$$E = \frac{b\,c \sin A}{2\,R^2 \sin 1''}$$

in which E is the spherical excess, b and c two sides of the plane triangle, A the included angle and R the mean radius of the earth. The expression, $\frac{1}{2}\,b\,c \sin A$ is the area of the plane triangle. It is evident that any other expression for the said area may be substituted for this one in the formula for spherical excess.

When no correction is made for spherical excess, the back-azimuth is given by the formula,

$$Z' = Z \pm 180°.$$

When, however, the correction for spherical excess is made, or when convergency is considered, the formula for back-azimuth becomes

$$Z' = Z \pm 180° \pm dZ$$

in which Z' is the back-azimuth, Z the forward-azimuth, and dZ the correction for azimuth. The upper signs are used when $Z < 180°$ and the lower when $Z > 180°$. The correction for azimuth, dZ, is computed by the usual L, M, Z formulas, as follows:

$$\text{Spherical excess } E = \frac{b\,c \sin A}{2R^2 \sin 1''}$$

R, mean radius of the earth = 6367395.6 meters.

$$\text{Factor } F = \frac{1}{2R^2 \sin 1''} \quad \text{Log. } F = 1.40547$$

Latitude, Longitude and Azimuth.

$$L - L' = + dL = BK \cos Z + CK^3 \sin^2 Z + Dk^3.$$

$$M - M' = - dM = AK \frac{\sin Z}{\cos L'}$$

$$Z + 180 - Z' = - dZ = dM \sin L_m.$$

To make these calculations I use printed blanks of the following form:

LATITUDE, LONGITUDE and AZIMUTH.

A Divisaderos B Prieto C Yaqui

A	68° 29′ 37″	68° 29′ 37.″25	a	22631.8	log	4.3547206
B	78 84 53	78 84 53, 25	b	23844.0	log	4.3778796
C	32 55 30	82 55 30, 25	c	18221.8	log	4.1212989
	180 00′ 00″	180° 00′ 00.″75				

Spherical Excess.

Fact. F	1.40547
b	+ 4.37738
c	+ 4.12129
sin A	+ 9.99866
	1.87280
E	0″.746
½ E	0″.248

True Azimuths (direct and back.) Az N.E.

Azimuth of Yaqui-Prieto	8° 09′ 06.″8
Spherical angle at Yaqui	+ 32 55 30. 3
Azimuth of Yaqui-Divisaderos Z =	41 04 37. 1
	180
Z + 180° =	221 04 37. 1
d Z =	+ 04 52. 8 +
Azimuth of Divisaderos-Yaqui Z_1 =	221° 09′ 29.″9

Geographical Coordinates.

Fact. B	8.5115729	+	Fact. C	1.16692	+	Fact. D	2.3294	+
K	+ 4.3773796	+	K^2	+ 8.75476	+	h$''$	+ 5.5324	+
cos. Z	+ 9.8772720	+	sin^2 Z	+ 9.63523	+			
	2.7662245	+		9.55691	+		7.8618	+
h	582″.74	+	h′	0″.36	+	h″	0″.007	+

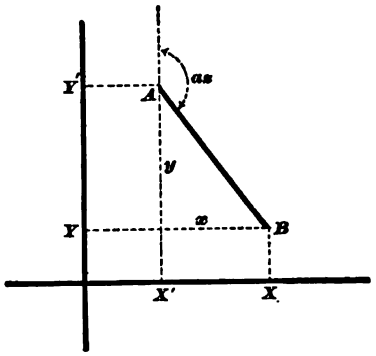
h	9° 43″.74	+	Fact. A	8.5093588	+	d M	2.76753	—
h′	+ 0 .36	+	K	+ 4.3773796	+	sin L$_m$	+ 9.69903	+
h″	+ 0 .01	+	sin Z	+ 9.8176127	+		2.46656	—
+d L	9′ 44″.11	+	cos. L′	— 9.9866184	+	−d Z	292″.79	—
				2.7675827	+			
			−d M	585″.50	+			

Coordinates of Divisaderos

L	30° 00′ 00″	M	31° 20′ 00″
d L	+ 9 44	d M	— 9 45.5
L′	30° 09′ 44″	M′	31° 10′ 14″.5
	Latitude N.		Longitude West of Greenwich.

After calculation of the triangulation, with correction for spherical excess, when necessary, and computation of the back-azimuth of every line, the rectangular co-ordinates of every point, referred to some suitable point as an origin, may be calculated as follows:

FIG. 2.



In Fig. 2, A is a point of which the co-ordinates X′ and Y′ are known; B is a point of which the co-ordinates X and Y are required. Obviously, $X = X' \pm x$; and $Y = Y' \pm y$; x and y being the departure and latitude of B from A. These departures and latitudes are given by the following formulas, u being the forward-azimuth and b the distance:

Quadrants.	Azimuths.	x	y
I	$u < 90°$	$+ b \sin u$	$+ b \cos u$
II	$u > 90°$	$+ b \sin (180° - u)$	$- b \cos (180° - u)$
III	$u > 180°$	$- b \sin (u - 180°)$	$- b \cos (u - 180°)$
IV	$u > 270°$	$- b \sin (360° - u)$	$+ b \cos (360° - u)$

The following is the form of blank employed:

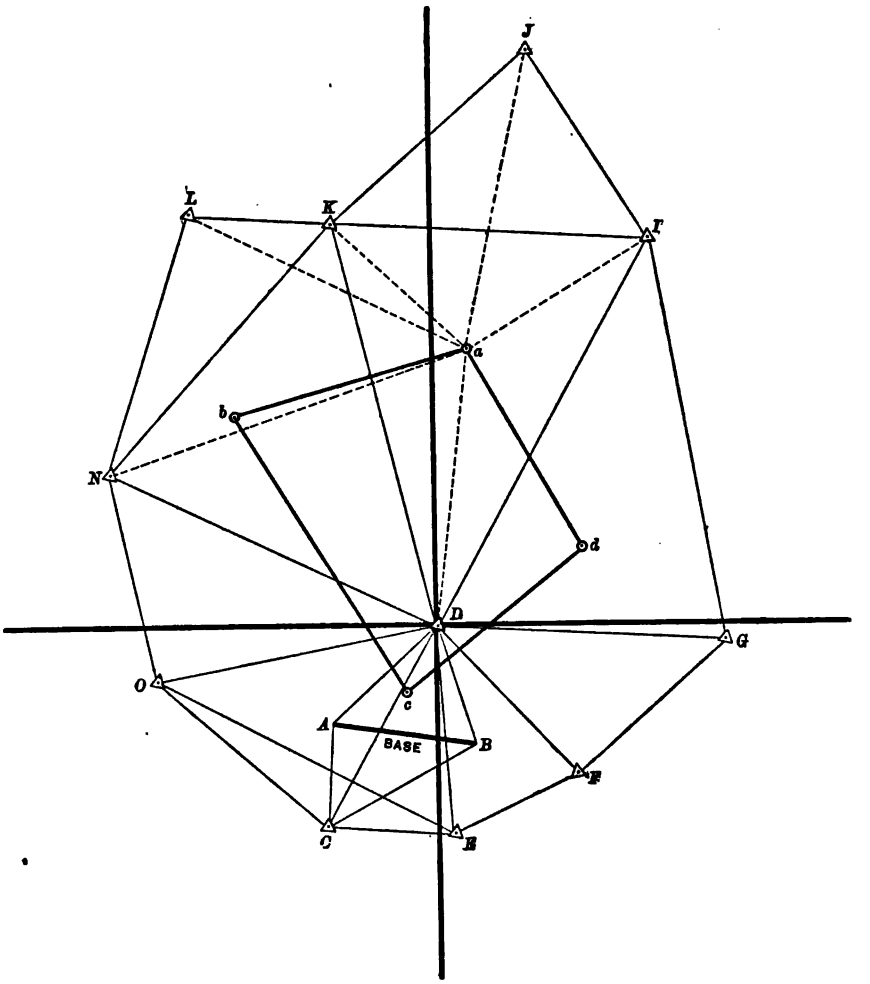
PROJECTIONS and COORDINATES.

A—D

AD			4.0866440		4.0866440
az=178° 40′ 51″	sin	+	8.8021887	cos	+ 9.999 8449 —
			2.8888327		4.0865289 —
	x		250.4 +	y	10887.5 —
A	X_1		33801.0 —	Y_1	47989.0 +
D	X		33550.6 —	Y	37111.5 +
az=	sin	+	—	cos	+ —
			—		—
	x		—	y	—
	X_1		—	Y_1	—
	X		—	Y	—

Filling In.—The most expeditious method of filling in is that of the "three-point problem." The practice is to go to every corner of private lands, every spring, every mountain peak not a triangulation-point, and every other point which it is desired to locate on the map, and take the azimuth of every flag that is visible, precisely as was done in the triangulation. Here also all that is wanted is the angle between every two lines. If it is desired to meander a stream or road or trail or locate important topographical features of the country, observations as described are taken on every flag that is visible. Each point may be calculated by several sets of three points and the results may be compared as a check. For instance, in the case shown by Fig. 3, the point a may be calculated from the points, IJK, or IJL, or IKN, or JKL, or KLN, or IKD, or KND, or LND, etc.

FIG. 3.



The three-point problem is calculated as follows:

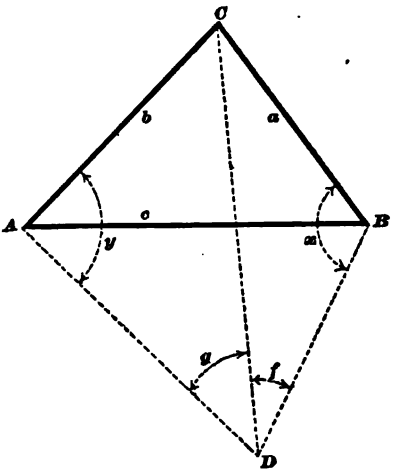
Case I.

The sides a, b and c, and their azimuths, the angles A, B and C, and the observed angles f and g, having been given by previous calculations or observations, it is required to find AD, or CD, or BD.

In the triangles BCD and ACD

$$\mathrm{CD} = \frac{a \sin x}{\sin f} = \frac{b \sin y}{\sin g}, \text{ whence } \qquad \frac{\sin y}{\sin x} = \frac{a \sin g}{b \sin f}.$$

FIG. 4.



Let m be any angle, such that

$$\tan m = \frac{a \sin g}{b \sin f} = \frac{\sin y}{\sin x}, \text{ whence } \sin y = \sin x \tan m. \quad \text{But}$$

$$\frac{\tan \frac{1}{2}(x-y)}{\tan \frac{1}{2}(x+y)} = \frac{\sin x - \sin y}{\sin x + \sin y} = \frac{\sin x - \sin x \tan m}{\sin x + \sin x \tan m}$$

$$= \frac{1 - \tan m}{1 + \tan m} = \cot (m + 45°)$$

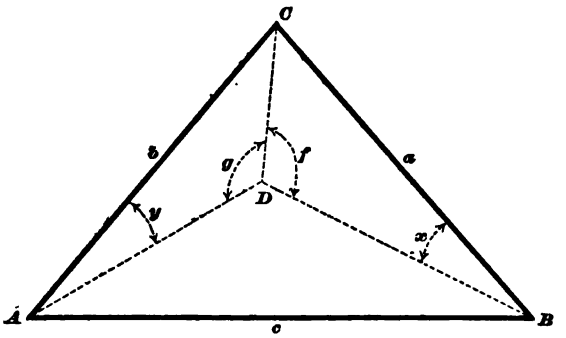
$$\tan \tfrac{1}{2}(x-y) = \tan \tfrac{1}{2}(x+y) \cot (m + 45°)$$
$$\tfrac{1}{2}(x+y) = 180° - \tfrac{1}{2}(f + g + C).$$

Having found x and y, we have, in the triangle ABD, the angle at B, or $B' = x - B$, and the angle at A, or $A' = y - A$; and AD or BD can be found by the usual sine-formula.

Case II.

The same elements being given and required, but D being within the triangle ABC, the calculation results as follows:

FIG. 5.

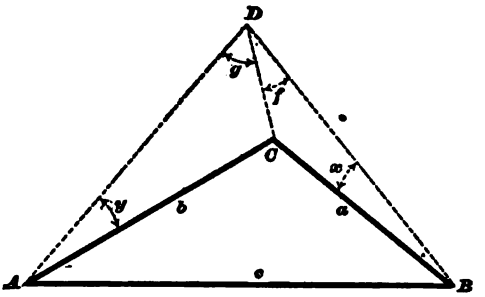


$$\tfrac{1}{2}(x+y) = 180° - \tfrac{1}{2}(f+g+C)$$
$$B' = B - x, \text{ and } A' = A - y.$$

Case III.

For this case, presenting a still different relative position of D, the result is as follows:

FIG. 6.



$$\tfrac{1}{2}(x+y) = \tfrac{1}{2}[C-(f+g)]$$
$$B' = B + x, \text{ and } A' = A + y.$$

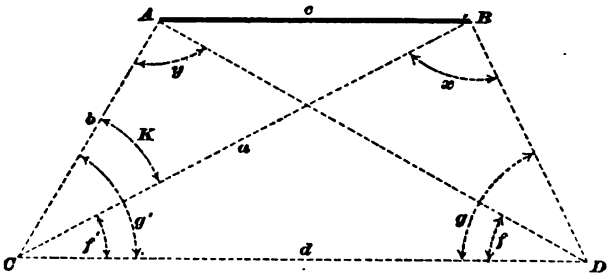
The correctness of the calculation may be verified in either of the three cases by the formula:

$$\tan m = \frac{BD \sin (y+g)}{AD \sin (x+f)}.$$

It sometimes happens that only one or two flags are visible, in which case auxiliary points may be used. For such a purpose, select a point from which at least three points are visible and which is it-

self visible from the point whose location is required. Locate the auxiliary point by the "three-point problem," and then the required point by means of the two visible flags and the auxiliary point just established. Or, when only two flags are visible, as A and B from C, in Fig. 7, we may proceed as follows:

FIG. 7.



At C take the azimuths of A and B and any third point as D, from which A, B, and C are visible. Go to D and take the azimuths of A, B, and C.

In the triangle BCD,

$$a = \frac{d \sin g}{\sin x}; \text{ and in ACD, } b = \frac{d \sin f}{\sin y}; \text{ whence}$$

$$\frac{b}{a} = \frac{d \sin f \sin x}{d \sin y \sin g} = \frac{\sin f \sin x}{\sin g \sin y}.$$

Let m be any angle such that

$$\tan m = \frac{b}{a} = \frac{\sin f \sin x}{\sin g \sin y} \text{ or } \cot m = \frac{a}{b}; \text{ then}$$

$$\frac{\cot m - 1}{\cot m + 1} = \frac{a - b}{a + b} = \cot (m + 45°).$$

In the triangle ABC,

$$\tan \tfrac{1}{2}(A - B) = \frac{a - b}{a + b} \tan \tfrac{1}{2}(A + B) = \tan \tfrac{1}{2}(A + B) \cot (m + 45°).$$

Since A + B = 180° − K, the values of A and B can be calculated; and in the same triangle, ABC,

$$b = \frac{c \sin B}{\sin K} \text{ and } a = \frac{c \sin A}{\sin K}$$

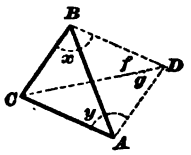
For these calculations, including those of the co-ordinates of the points thus determined, printed blanks of the following form are used:

THREE POINTS.

F.B.No. 6 D Pitiquito Az N.W.

A Chanate B Salada C Carnero

C=AB=	25796.0	log	4.4115527	az =	163° 51′ 02″
a=BC=	20085.8	log	4.3028901	az =	287 52 39
b=AC=	22113.3	log	4.3446538	az =	212 40 59
A	45° 49′ 57″	B	55° 56′ 28″	C	75° 11′ 40″

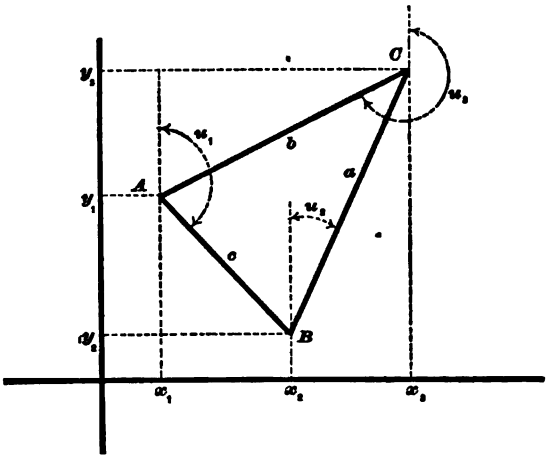


f=66° 25′ 45″
g=69 09
f+g=135 34 45
C+f+g=210 46 25
½(C+f+g)=105 23 13
½(x+y)=180°−½(C+f+g)=74 36 47

a	4.3028901	m+45° =	87° 48′ 10″	cot	8.5830643 +
sin g	+9.9705865	½(x+y)=	74 36 47	tan	+10.5603504 +
	4.2734766	½(x−y)=	7 56 18 +	tan	9.1433147 +
b	−4.3446538				
sin f	−9.9621639	x =	82 33 00	x−B	26° 34′ 37″
tan m	9.9666589	y =	66 40 34	y−A	17 50 37
m	=42° 48′ 10″				
c	4.4115527				
sin(f+g)	−9.8450503			a	
	4.5665024		4.5665024	sin f	—
sin(x−B)	+9.6506952	sin(y−A)	+9.4863170	sin x	+
	4.2171976		4.0528194		
A D	16849.1	B D	11298.2	C D	
az AC =	212° 40′ 59″	A D	4.2171976		4.2171976
y =	−66 40 34	sin az AD	+9.7474886 +	cos az AD	+ 9.9186097 −
az AD =	146 00 25		3.9646812 +		4.1358073 −
		x	9219.0 +	y	13871.2 −
		A X =	6871.0 +	Y	= 10849.0 +
		D X$_1$ =	16090.0 +	Y$_1$	= 2822.2 −

In extended calculations it frequently happens that the three points are given by their co-ordinates alone. It then becomes necessary to calculate the triangle, which is done by the following formulas:

FIG. 8.



$$\tan u_1 = \frac{x_2 - x_1}{y_2 - y_1} \qquad c = \frac{x_2 - x_1}{\sin u_1} = \frac{y_2 - y_1}{\cos u_1}$$

$$\tan u_2 = \frac{x_3 - x_2}{y_3 - y_2} \qquad a = \frac{x_3 - x_2}{\sin u_2} = \frac{y_3 - y_2}{\cos u_2}$$

$$\tan u_3 = \frac{x_3 - x_1}{y_3 - y_1} \qquad b = \frac{x_3 - x_1}{\sin u_3} = \frac{y_3 - y_1}{\cos u_3}$$

$$A = az.AB - az.AC: \; B = 360° - (az.BA - az.BC): \; C = az.CA - az.CB.$$

In these calculations great care must be had in the use of the signs, in order to get the correct values of u_1, u_2, and u_3.

If $x_2 - x_1$ is + and $y_2 - y_1$ is +, tan u_1 is +; $u_1 < 90°$ and is in quadrant I.

If $x_2 - x_1$ is + and $y_2 - y_1$ is —, tan u_1 is —; $u_1 > 90° < 180°$ and is in quadrant II.

If $x_2 - x_1$ is — and $y_2 - y_1$ is —, tan u_1 is +; $u_1 > 180° < 270°$ and is in quadrant III.

If $x_2 - x_1$ is — and $y_2 - y_1$ is +, tan u_1 is —; $u_1 > 270° < 360°$ and is in quadrant IV.

If the bearings are wanted, no attention need be given to the signs, as they may be taken directly from the tables. Or the azimuths may be converted into bearings and *vice versa*. If

Azimuth $< 90°$, Bearing = Azimuth and is N. E.

Azimuth $> 90° < 180°$, Bearing = 180° — Azimuth and is S. E.

Azimuth > 180° < 270°, Bearing = Azimuth — 180° and is S. W.
Azimuth > 270° < 360°, Bearing = 360° — Azimuth and is N. W.
The printed blanks for these calculations are of the following form:

TRIANGLES.
by Coordinates.

Az. N.W.

A Chanate		B Salada		C Carnero	
A, x_1 =	6871.0 +	$x_2 - x_1$ =	7175.0 +	y_1 =	10849.0 +
B, x_2 =	14046.0 +	$x_3 - x_2$ =	19116.0 −	y_2 =	13929.0 −
C, x_3 =	5070.0 −	$x_3 - x_1$ =	11941.0 −	y_3 =	7763.0 −
$y_2 - y_1$	6.8556219	+	6.8556219		
$y_3 - y_1$	4.3940662	−	sin u_1	9.4442692	
tan u_1	9.4617557	−		4.4115527	
az A B = u_1 =	165° 51′ 02″		c = A B =	25796.0	
$x_2 - x_3$	4.2813970	−		4.2813970	
$y_2 - y_3$	3.7900085	+	sin u_2	9.9785069	
tan u_2	10.4913935	−		4.3028901	
az B C = u_2 =	287° 52′ 39″		a = B C =	20085.8	
$x_3 - x_1$	4.0770407	−		4.0770407	
$y_3 - y_1$	4.2697930	−	sin u_3	9.7333869	
tan u_3	9.8072477	+		4.3440538	
az A C = u_3 =	212° 40′ 59″		b = A C =	22113.3	

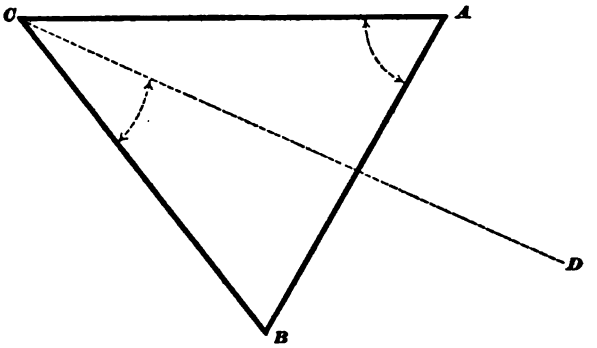
A		B		C	
A, x_1 =		$x_2 - x_1$ =		y_1 =	
B, x_2 =		$x_3 - x_2$ =		y_2 =	
C, x_3 =		$x_3 - x_1$ =		y_3 =	
$x_2 - x_1$					
$y_2 - y_1$		sin u_1			
tan u_1					
az A B = u_1 =		c = A B =			
$x_2 - x_3$					
$y_2 - y_3$		sin u_2			
tan u_2					
az B C = u_2 =		a = B C =			
$x_3 - x_1$					
$y_3 - y_1$		sin u_3			
tan u_3					
az A C = u_3 =		b = A C =			

Unimportant Points.—Unimportant points may be located by simple intersection, thus:

Given (Fig. 9) AC and its azimuth, and the angle A, to find the azimuths of AB and CB, solve the triangle and locate B.

It is necessary to use a third point, D, visible from C, and whose azimuth CD is known, as also the observed angle between CD and

FIG. 9.



CB. From the known azimuth of AC and the angle CAB, the azimuth of AB is found, and from the azimuth of CD and the angle BCD the azimuth of CB is found. From the azimuths of the three sides the interior angles can easily be found, as follows:

Sides.	Angles.	Azimuths.
AC	.....	106° 10′ 13″
AB	115° 45′ 07.5″	221° 55′ 20.5″
CD	.....	269° 01′ 32″
CB	7° 30′ 04″	261° 29′ 28″
CA	24° 40′ 45″	286° 10′ 13″
BA	.....	41° 55′ 20.5″
BC	39° 34′ 07.5″	81° 29′ 28″

That is, the measured angle at A, 115° 45′ 07.5″, between B and C, added to the known azimuth of AC, 106° 10′ 13″, gives the azimuth of AB, 221° 55′ 20.5″.

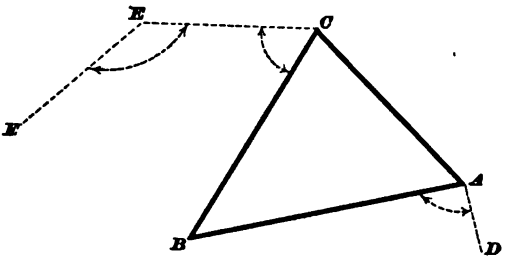
The measured angle C, 7° 30′ 04″, between B and D, subtracted from the known azimuth of CD, 269° 01′ 31″, gives the azimuth of CB, 261° 29′ 28″. This latter subtracted from the azimuth of CA, or the back-azimuth of AC, 106° 10′ 13″ + 180° = 286° 10′ 13″, gives the angle C, 24° 40′ 45″, between A and B.

The azimuth of BC, or the back-azimuth of CB, 261° 29′ 28″ — 180° = 81° 29′ 28″, less the azimuth of BA, or the back-azimuth of AB, 221° 55′ 20.5″ — 180° = 41° 55′ 20.5″ gives 39° 34′ 07.5″, the angle at B between A and C.

With the side AC and the three angles, the triangle can be readily solved.

At times the problem is presented in this form.

FIG. 10.



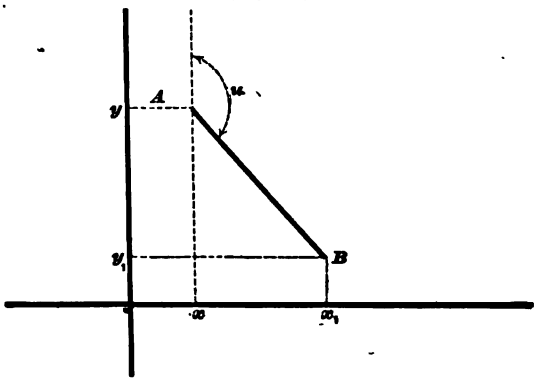
Given AC and its azimuth, required the position of the point B.

The azimuth of AD and the observed angle BAD are known, hence the azimuth of AB. The observed angle BCE is known, but not the azimuth of CE. The azimuth of EF and the angle FEC are known, hence the azimuth of EC. From the back-azimuth of EC, or the azimuth of CE and the angle BCE, the azimuth of CB is found. The azimuths of the three sides now being known, the angles are readily found and the triangle is solved. The form of the printed blanks for angles and azimuths has been already given.

With the co-ordinates of every point correctly calculated the map can now be plotted with the greatest facility and far more correctly than with protractor and scale.

The courses and distances may be calculated from the co-ordinates as follows:

FIG. 11.



$$\tan u = \frac{x_1 - x}{y_1 - y}$$

$$d = AB = \frac{x_1 - x}{\sin u} = \frac{y_1 - y}{\cos u}$$

The printed blank for these calculations is in the following form

COURSES and DISTANCES.

Salada	$x_1 =$	14046.0	+	$y_1 =$	18929.0	−
Chanate	$x =$	6871.0	+	$y =$	10849.0	+
	$x_1 - x =$	7175.0	+	$y_1 - y =$	24778.0	−
	$x_1 - x$	8.8558219	+		8.8558219	
	$y_1 - y$	4.3940662	−	sin u	9.4442692	
	tan u	9.4617557	−		4.4115527	
Chanate-Salada	$u =$	16° 06′ 58″ S. W.		$d =$	25796.0	
	$x_1 =$		−	$y_1 =$		−
	$x =$		−	$y =$		−
	$x_1 - x =$		−	$y_1 - y =$		−
	$x_1 - x$		−			
	$y_1 - y$		−	sin u		
	tan u		−			
	$u =$			$d =$		

Areas.—Areas may be calculated from the co-ordinates by the formula,

$$\text{Area} = \tfrac{1}{2}[y_1(x_5 - x_2) + y_2(x_1 - x_3) + y_3(x_2 - x_4) + y_4(x_3 - x_5) + y_5(x_4 - x_1)]$$

or

$$\text{Area} = \tfrac{1}{2}[x_1(y_5 - y_2) + x_2(y_1 - y_3) + x_3(y_2 - y_4) + x_4(y_3 - y_5) + x_5(y_4 - y_1)]$$

The form of reduction is as follows :

Corner.	Ordinates. (Y.)	Abscisse. (X.)	Difference of Alternate Abscisse.	Double Areas.
A	482	200	− 368	− 1773
B	319	718	− 831	− 2651
C	640	1031	− 63	− 403
D	892	781	+ 681	+ 6074
E	1210	350	+ 581	+ 7030

Plus area..... + 13,104
Minus area..... − 4,827

2)8,277

4,138 sq. met'rs.

That is, 350 — 718 = —368; 200 — 1031 = — 831; 718 — 781 = — 63; 1031 — 350 = + 681; and 781 — 200 = + 581.

Mr. Flipper prefers to calculate areas by a modification of the well-known method of Double Meridian Distances. The following example will show the points of the method. The point or corner nearest the meridian passing through the center of the tract is selected as the point of beginning.

Courses.	Dist.	Projections.				D. Distances		Multl.	Brg.	Double Areas.	
		N.	S.	E.	W.	E.	W.			N. E. & S. W.	N. W. & S. E.
S. 72° 31′ 58″ E.	543	.....	163	518	.....	368	.....	0	...	.....	.....
N. 44° 16′ 37″ E.	448	321	.....	313	.....	681	.....	681	E.	2667	.....
N. 44° 46′ 18″ W.	355	252	.....	.....	250	63	.....	894	E.	2253	.....
N. 58° 34′ 46″ W.	536	318	.....	.....	431	.....	681	213	E.	677	.....
S. 11° 38′ 33″ W.	743	.....	728	.....	150	.....	581	368	W.	2679	.....
	.....	891	891	831	831	1262	1262	.....	...	.....	.....

2)8276

4138 sq. meters.

This is the same tract, the area of which is calculated above from the co-ordinates of the corners. The result is the same, of course.

It will be observed that the sum of the double distances east is equal to the sum of the double distances west; that the last multiplier is equal to the first double distance, with a contrary sign; and that the first multiplier is always zero.

1st D. D. = 1st E. or W. ± Last E. or W.

2d D. D. = 1st E. or W. ∓ 2d E. or W.

3d D. D. = 2d E. or W. ± 3d E. or W., etc.

2d Multiplier = 2d D. D.

3d Multiplier = 2d Multiplier ± 3d D. D., etc.

Last Multiplier = 1st D. D.

1st Multiplier = Last Multiplier ± 1st D. D. = 0.

The northings and southings are multiplied by the multipliers, and the results placed in the proper columns. The difference between the sums of the two columns is the double area, and half of this is the area required.

The direct method by co-ordinates is faulty in that there is no check to the accuracy of the work. The survey should first be balanced, and then the area may be calculated by either method. Having, however,

balanced the survey, the latter method is the simpler and shorter. By balancing the survey, errors in the co-ordinates may be discovered which probably would otherwise escape detection.

The projections, or latitudes and departures, may be calculated on the printed blank for courses and distances.

Advantages.—The advantages claimed for this system are:

1. The superior accuracy of surveys thus made as compared with those made by chain and compass, or chain and transit, commonly known as "traverse-surveying."

2. Its cheapness. The work is done far more rapidly and with a much smaller number of assistants than by any other method, thus reducing expenses both as to number of employees and as to time consumed.

3. Facility and accuracy in plotting.

4. The certainty that subsequent surveys of adjoining tracts made in the same manner will fit without the necessity of "fudging."

5. Absolute certainty of description in conveyances.

6. Simple and convenient notation and consequent facility of calculation.

7. The combinations of geodesy with ordinary surveying, so that latitudes, longitudes, and azimuths may be readily computed and large areas plotted on a polyconic or other projection. Where the system can be used in connection with accurately established geodetic points, as in the United States, a very high degree of accuracy may be obtained.

NOTES ON A NOVEL CABLE-TRANSFER FOR RAILROAD-CARS, AND THE USE OF THE PATENT LOCKED-WIRE ROPE.

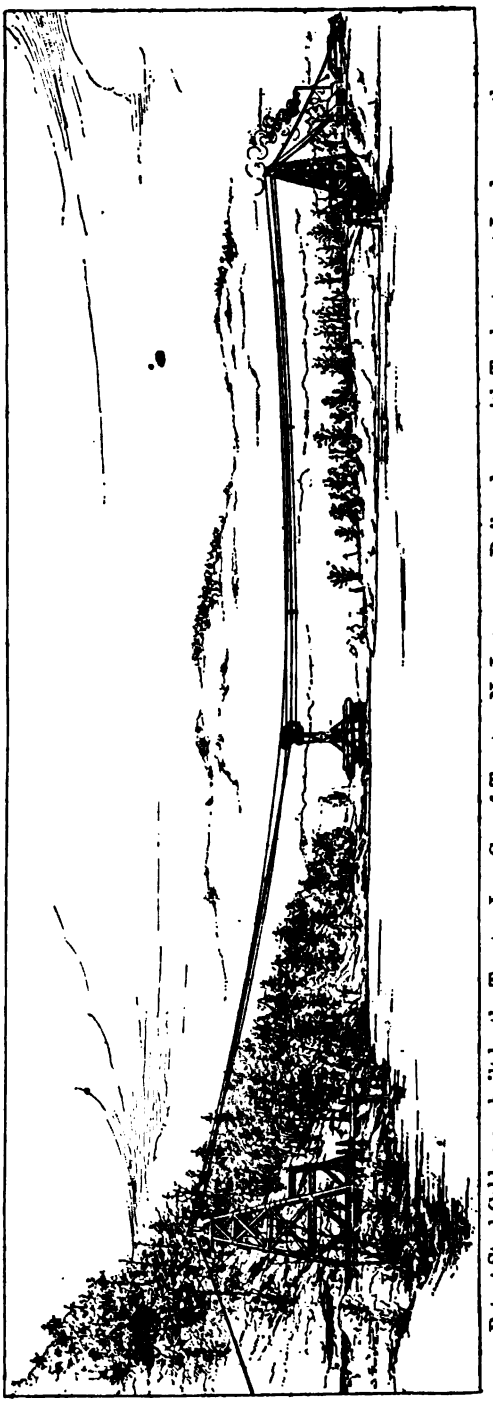
BY E. G. SPILSBURY, TRENTON, N. J.

(Glen Summit Meeting, October, 1891.)

As a fresh illustration of the advances made in the past few years in the overhead-cable systems of hoisting and conveying, as applied to open-cut deep mining and quarrying, I would call attention to a further development of this system of transportation which has lately been put in operation in this State. A few miles above Williamsport, on the Susquehanna river, the Glen Union Lumber Company, of Pottsville, Pa., owned a large tract of valuable timber-

land, which they proposed to bring into the market. Unfortunately, this land is situated on the right bank of the river, while the railroad, the only available means of marketing the lumber, runs on the left bank. The river is about 750 feet wide at this place, and the cost of a bridge to carry the cars over was practically prohibitory, as it threatened to consume the profits of the business for many years. Several plans for obviating the difficulty, such as a ferry-transport, or a floating bridge, were proposed; but these were considered impracticable by reason of the rapid floods and ice-gorges to which this river is subject. The problem was then submitted to the writer, who proposed as a solution, and subsequently erected, one of the Trenton Iron Company's cable-hoists. This transfer has now been in successful operation for more than a year, and has thoroughly demonstrated the feasibility of the system for carrying loads of 20 or 30 tons over very long spans.

The work to be done was to take the loaded railroad-cars from the tracks which brought them down to the river-bank, and deliver them without unloading to the tracks of the Philadelphia and Erie railroad on the opposite shore. The shortest span which could be obtained was 733 feet. The accompanying sketch shows the general construction. Two 70-foot towers were erected, one on each bank, and over the top of these were stretched two steel cables, each 2 inches in diameter, each cable having a breaking-strain of about 125 tons. In order to equalize the strains on these cables, and so keep them always parallel, they were made in one continuous length, the two loose ends being anchored rigidly on the railroad side of the river, and the bight at the other end passing round a 6-foot sheave, revolving in a shackle attached to the anchorage on the other bank. On these cables runs a carriage supported on four wheels, and moved back and forth by means of an endless cable 1 inch in diameter. In the lower part of this carriage are placed the hoisting-sheaves, which, in conjunction with a large fall-block, hoist the load vertically, high enough to free it from the tracks at each end and from the river at its highest stage in the middle. A wrought-iron lattice-work cage, 36 feet in length and fitted with tracks, receives the car at each end. The sag of the hauling-ropes and also of the hoisting-rope, which on so long a span would be very great, is taken up by a number of fall-rope carriers of the Locke patent design. Indeed, the whole arrangement of carriage and fall-blocks is only a modification of Locke's system to meet the special requirements of the case. The power required to operate the system is furnished by a 50-horse-power,



Patent Steel Cable-way, built by the Trenton Iron Co., of Trenton, N. J., to carry Railroad-cars with Twelve-ton net Loads across the Susquehanna River at Glen Union, Pa. Clear span, 735 feet.

specially-designed engine, built by the Lidgerwood Manufacturing Company. The total load carried is about 26 tons, including the cage and car; the estimated load of lumber being 12 tons. The trip across the river is made in about three minutes. The total cost of the whole structure completed was less than one-fifth the cost of a bridge, even of the lightest design.

During its operation last winter and spring, no delays occurred from ice-gorges or freshets. After it had been running for some months, however, it was found that the wear on the tread of the carriage-wheels was very great. This was obviously due to the uneven surface of the ropes, which tended to cut a spiral thread on the grooves corresponding to the lay of the strands in the rope. Of course, it would not do to make these wheels of harder material than the rope, since it is cheaper to replace them than it would be to wear out the rope; but, at the same time, the cost of replacing the wheels threatened to lessen greatly the economy of the whole system. Hence, it has now been determined to replace the two main cables of the ordinary make by two of the smooth-coil Elliot locked-rope cables, the use of which entirely obviates all this wear on the wheels. These ropes are now being manufactured, and I hope to present, at some future meeting, figures thoroughly demonstrating the great advantages of these ropes over all others for just such purposes.

It may be permissible to say here a few words about the progress of the introduction of this Elliot rope in this country. When we first began to manufacture it, we supposed that all the difficulties likely to arise had been met by the patentees in their English works; but we soon found that such was not the case, and that there was much to learn before we could make it a thorough success. As many of the members of the Institute are aware, several of the first ropes of this kind which we placed in collieries of the anthracite region did not give long enough extra service to warrant their extra cost. This disappointment was due, in large part, to a lack of experience on our part as to the exact quality of steel necessary for the ropes, and also to the fact, which we have since found out, that these ropes must be designed specially for the use they are to be put to, and ought not to be used indiscriminately from stock, as is often the case with the ordinary ropes. Wherever the conditions have been thoroughly made known to the manufacturer beforehand, these ropes are now giving the utmost satisfaction, and I feel sure that gradually, as they become better known, their use will be very greatly extended. In the English collieries to-day, their use is increasing rapidly, not-

withstanding all the prejudices which they have had to combat from other manufacturers, and from the defects which many of the first ropes manifested. A leading colliery-owner in Wales made the remark to Mr. Abram S. Hewitt last summer, when the latter was in England, that notwithstanding their aversion to the change, the increased degree of safety to their men's lives which these locked ropes offered was so great that they could not now afford to use any other kind.

In this country, the managers of western mines seem to appreciate the economy of the locked rope; and the demand for it from that section is increasing considerably. In closing I merely call attention to two cases in which the Elliot rope is giving remarkable results:

The Gauley Mountain Coal Company, of West Virginia, put in, in the spring of 1890, two of these ropes of 1 inch diameter on its plane, which is 2500 feet in length; the "trips" consisting of two cars, each carrying about 6 tons. Up to last month, over 200,000 tons of coal had been run down; and at the time I saw the cables, a few weeks since, there was not the slightest sign of wear to be seen anywhere on them. It was impossible to note with callipers any lessening of the diameter; notwithstanding these ropes had been subjected to excessive strains, owing to two accidents on the plane, in one of which one of the ropes had been torn out of the "barney" by a collision. The Manager, Mr. Wm. N. Page, informed me also that since the introduction of these ropes he had not had to replace one single track-roller, and that they were still as good now as the day they were put in, over twenty months ago.

The other case is a locked rope, 1¼ inch diameter and 4000 feet long, which has now been running over a year at the Central Copper Mine in Michigan. This rope hoists a load of about 9 tons, at an average speed of 2900 feet a minute. It has already outlived any ordinary rope used heretofore in this place, and from a report received a few weeks since, it shows absolutely no sign of wear whatever. The Superintendent further reports that, whereas with the ordinary rope they wear out the grooves of their head-sheave every three months, they have not had to replace this sheave since the locked rope was put in, and it is as smooth now as when new.

The Northwest Coal Company, of Scranton, has also used our locked ropes with much success. In a letter received from them, after detailing the work done by these ropes, they sum up as follows: "The result has been very satisfactory; in the first place the 1-inch

patent locked rope did the work of a 1¼-inch ordinary rope and lasted at least twice as long as the average life of an ordinary wire rope. We regard it as the most economical rope in use, and indeed are very well satisfied with it."

As is the case with every other novelty, it will take time to prove the advantages of this rope. It seems to me, however, that the locked rope has come to stay, and while it is not adapted to every use the ordinary rope can be applied to, still I am inclined to think that for mining purposes it will soon be much more sought after and relied upon than it has been heretofore.

TESTS AND REQUIREMENTS OF STRUCTURAL WROUGHT-IRON AND STEEL.

(Discussion continued from page 731.)

GUSTAVUS C. HENNING, New York City: I desire to correct an injustice inadvertently committed in my contribution (pp. 715 to 726) to this discussion. Namely, on page 717, line 1, after the words, "cannot be filled in the East," I wish to add the words, "except by the Catasauqua Manufacturing Co." The same qualification should be made in the similar statement on page 718, line 8; and on page 726, line 22, the words "by the Catasauqua Manufacturing Co.," should be introduced before the words, "in the Lehigh Valley."

THE ALLUVIAL TIN-DEPOSITS OF SIAK, SUMATRA.

SECRETARY'S NOTE: The paper of Mr. Rolker on the above subject, printed on pp. 50 to 84 of the present volume, was necessarily passed through all stages of publication without revision from the author, who was during the whole period in the interior of Africa. Some errors, especially in the spelling of proper names and foreign words, naturally occurred in spite of all care taken to prevent them; and the following corrections are therefore required:

Page 52, line 19 from top, Kanau should be Kanan.

Page 52, line 20 from top, Kanau should be Kanan.

Page 52, line 23 from top, Kanau should be Kanan.

Page 52, line 24 from top, Kanau should be Kanan.
Page 52, line 24 from top, Podang should be Padang.
Page 54, line 13 from bottom, Kanau should be Kanan.
Page 60, line 5 from bottom, Pingir should be Pingi.
Page 60, line 4 from bottom, Bleoon should be Blevon.
Page 61, line 20 from bottom, *area* should be *arca*.
Line 8 from the bottom on page 82 is out of place. It belongs under the table on page 83, upon which it is a comment. And the title of this table should show that it is taken from mercantile sources.—R. W. R.

THE BENDIGO GOLD-FIELD.

SECRETARY'S NOTE: In the paper on this subject by Mr. Rickard (page 463 of this volume), mention is made (on page 526) of Mr. E. J. Dunn as having been the first to point out the true relation of the Bendigo deposits to the stratification of the enclosing rocks; and reference is made to his note on the subject, in the Quarterly Report of the Mining Department of Victoria for December, 1888, concerning which Mr. Rickard says (page 527): "Had it come under my notice previous to going to Sandhurst, I should have been spared much of the difficulty which I experienced before the true nature of the deposits became manifest."

Mr. Dunn is not satisfied with this acknowledgment, and has addressed to me a letter, claiming that he personally furnished to Mr. Rickard much information as to the series of anticlinals, the true nature of the saddle-reefs, the difference between the bedding and the cleavage-planes, etc.; that he explained the problem to Mr. Rickard, and corrected the views of the latter; and that the records and plans of the Underground Survey Office were also placed, by the late E. T. Brown, at Mr. Rickard's disposal, a service which should have been, but was not, acknowledged in the paper.

Mr. Rickard informs me that these statements are well-founded, and requests me to disavow on his part all intention to withhold from Mr. Dunn the credit he deserves. This disclaimer is made by me, instead of Mr. Rickard, because I am responsible for the final form of his paper, which I edited for publication, and (in view of its great length) freely condensed. Exactly what I cut out as

superfluous, I cannot now recall; but I remember that the manuscript was filled to an extraordinary extent with personal acknowledgments, and that I thought *one apiece* was enough. I confess, it still appears to me, that, after the plain mention of Mr. Dunn as "a noteworthy exception" to writers on Bendigo, in having published the first correct statement of the relations of its gold-deposits, it was scarcely necessary to add that he had also made the same statement verbally, at a later day. The greater credit includes the less. If, acting under that impression, I omitted phrases from Mr. Rickard's manuscript which would have pleased Mr. Dunn, I accept the blame, and trust he will accept this reparation.—R. W. R.

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